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User's Manual for Premining Planning of Eastern Surface Coal Mining. Volume 2 Surface Mine Engineering

Pennsylvania State Univ., University Park

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16. ABSTRACT

This document is the second in a series of six volumes which together comprise a User's Manual for Premining Planning of Surface Coal Mining Operations in the Eastern United States. Recommended methods, techniques and alternatives for selecting and designing mining systems, based on a review and critical evaluation of the methods reported in the literature and applied in the field, and using the geological and hydrological settings prior to mining as basic inputs, have been structured into guidelines for developing, evaluating and selecting mining and reclamation plans, prior to commencement of mining, that will be least detrimental to the environment.

This report addresses the surface mine engineering aspects. A comprehensive summary of the analyses required to layout a surface mine, taking into consideration the limitations imposed by geology, equipment, reclamation, economics, and environmental control requirements is provided. Methods, techniques, and alternatives are recommended for selecting and designing mining systems that include soil handling and storage, overburden removal, and minimization of environmental effects of drilling, blasting, off-highway transportation, coal loading, and steep slopes.

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Geology	Engineering	Eastern Coal Regions	
Hydrology	Blasting	Reclamation Planning	
Water Quality	Exploration	Environmental Planning	
Reclamation	Slope Stability	Land Use Planning	
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VOLUME 2

SURFACE MINE ENGINEERING

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FOREWORD

When energy and material resources are extracted, processed, converted, and used, the related pollutional impacts on our environment and even on our health often require that new and increasingly more efficient pollution control methods be used. The Industrial Environmental Research Laboratory-Cincinnati (IERL-Ci) assists in developing and demonstrating new and improved methodologies that will meet these needs both efficiently and economically.

This document is the second in a series of six reports designed to provide the surface coal mining industry and its regulators with a comprehensive review of the best available methods for extracting coal while protecting the environment. This volume summarizes the engineering and economic analyses that are necessary for premining planning of surface coal mining operations in the eastern coal fields of the United States. For further information on this series of reports, contact the Energy Pollution Control Division.

David G. Stephan
Director
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ABSTRACT

The purpose of this research effort was to study the surface mining of coal in the Eastern United States and to establish guidelines for developing, evaluating, and selecting the least environmentally detrimental mining and reclamation practices. The study was to consider the geological and hydrological settings before mining as the basic inputs to premining planning, and guidelines were to be developed for assessing alternatives in the areas of surface mine engineering, water management, and land use planning.

The recommended methods, techniques, and alternatives for selecting and designing mining systems are based on a review and critical evaluation of the methods reported in the literature and applied in the field. They are presented in a series of six reports, which together make up a user's manual for premining planning of surface coal mining operations in the Eastern United States.

This report, which is the second in a series of six, addresses the surface mine engineering aspects. A comprehensive summary of the analyses required to layout a surface mine, taking into consideration the limitations imposed by geology, equipment, reclamation, economics, and environmental control requirements is provided. Methods, techniques, and alternatives are recommended for selecting and designing mining systems that include soil handling and storage, overburden removal, and minimization of environmental effects of drilling, blasting, off-highway transportation, coal loading, and steep slopes.

This report was submitted in partial fulfillment of Grant No. R803882 by the Department of Mineral Engineering of The Pennsylvania State University under the sponsorship of the U.S. Environmental Protection Agency. This report covers the period July 1, 1975, to May 30, 1978, and work was completed as of June 1, 1978.

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PROJECT STAFF

The materials contained in the manual were prepared by an interdepartmental and interdisciplinary group of the College of Earth and Mineral Sciences of The Pennsylvania State University. Overall management for the project was provided by the Department of Mineral Engineering. The project staff was comprised of the following personnel:

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CONVERSION FACTORS

In this report, wherever practicable, the data are provided in metric units. However, in many instances, particularly in figures and tables, the units used are those that are commonly designated and understood by the personnel in the mining industry. Conversion to metric units, in these cases, may not only be not useful but frustrates the reader with unfamiliar units. Therefore, this conversion table is provided for use with this report.

Mass
1 short ton = 0.907 metric ton
1 long ton = 1.016 metric ton
1 pound = 0.4536 kilogram

Length
1 inch = 2.54 centimeters
1 foot = 30.48 centimeters
1 yard = 0.9144 meter
1 mile = 1.609 kilometers

Square Measure
1 square inch = 6.452 square centimeters
1 square foot = 0.0929 square meters
1 square yard = 0.836 square meters
1 acre = 0.405 hectare

Cubic Measure
1 cubic foot = 0.0283 cubic meters
1 cubic yard = 0.0765 cubic meters

Liquid Measure
1 gallon = 3.785 litres

Density
1 pound (mass)/cubic foot = 0.01602 gram/cubic centimeter

Force
1 pound (force) = 0.4536 kilogram force
1 pound (force) = 4.448 newtons

Energy
1 British Thermal Unit = 1055 Joules
1 foot pound (force) = 1.356 Joules

Power
1 horsepower = 745.7 watts

Pressure
1 pound (force) per square inch = 6895 newtons per square meter

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SECTION 1

INTRODUCTION

GENERAL

By the year 2000 A.D., the annual U.S. demand for energy is expected to double and that for the world, to triple (Starr, 1971). Gordon (1976) has summarized a sample of forecasts of U.S. energy consumption for the years 1985 and 2000 from several sources and task forces dealing with various aspects of the appraisal (Table 1). These projected increases will try man's ability to explore, discover, extract, and beneficiate the fuels in necessary volumes, to transport them safely, and to dispose of effluent wastes with minimum harmful environmental effects. Considering the difficulties experienced at present in mining without environmental damage, in finding acceptable locations for new power plants, and in controlling stack emissions from existing plants, the energy projections for the year 2000 indicate the need for a thorough evaluation of the available options and cautious planning.

ENERGY FORMS

In order to meet the growing demand for energy, all fuels will be required in increasing quantities. Economics and technology must also assure that each fuel is put to the use for which it is best suited. Coal has gradually overtaken natural gas and petroleum in the field of electricity generation, and the use of nuclear energy has increased. Over a longer period of time, nuclear energy may replace the fossil fuels (coal, oil shales, tar sands, natural gas, and petroleum) as the leading fuel in electricity generation, provided that technological advancements occur as predicted. The use of geothermal, solar radiation, and tidal energies is not yet widely practical and/or economical. However, coal, particularly surface-mined coal, may compete with nuclear energy for some time. Forecasts indicate that as technology in the field of coal gasification progresses, demand for coal may shift from electricity generation to synthetic gas and liquefied products. This would tend to remove some of the burden from natural gas and oil and extend the projected peaks of each fuel. If coal gasification becomes feasible by the early 1980's, the demand for coal may accelerate more rapidly than predicted. Although there is no exact accuracy in forecasting energy demand and fuel usage, coal should have an assured position throughout the future because of its abundant widespread distribution and chemical versatility. These forecasts are made in a complex ever-changing world where much depends upon international politics, technological innovation, and economic decisions. A projection of energy or

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TABLE 1. SELECTED FORECASTS OF TOTAL U.S. ENERGY CONSUMPTION IN 1985 AND 2000 (QUADRILLION BTU) (GORDON, 1976)

		1985	2000
Dupree and West		116.63	191.90
National Petroleum Council --	Low	112.50	n.a.
	High	124.90	n.a.
	Intermediate	130.00	n.a.
Project Independence --	BAU* $ 7 Oil (no conservation)	109.06	n.a.
	AS* $11 Oil (with conservation)	96.25	n.a.
Energy Policy Project --	Historic Growth	116.00	187.00
	Technical Fix	92.00	124.00
	Zero Growth	88.00	100.00
Organization for Economic Cooperation and Development --	Base	114.32	n.a.
	$ 6 Oil	105.83	n.a.
	$11 Oil	100.09	n.a.
ERDA* --	No new initiatives	107.30	165.47
	Conservation	96.97	122.48
	Synthetic fuels	107.28	165.42
	Electrification	106.77	161.16
	Limited nuclear	107.05	158.01
	Conservation, Synthetic fuels and Nuclear	98.14	137.03
EEI* --	High growth, Low energy prices	124.20	193.93
	High growth, High energy prices	112.90	179.10
	Medium growth, High energy prices	104.30	155.10
	Low growth, High energy prices	98.90	104.50
Dupree and Corsentino		103.54	163.43
FEA* --	1976 Reference $ 8 Oil	103.29	n.a.
	1976 Conservation $16 Oil	92.51	n.a.

BAU: Business as Usual.
AS: Accelerated Growth Scenario.
ERDA: Energy Research and Development Administration.
EEI: Edison Electric Institute.
FEA: Federal Energy Administration.
1973 Energy consumption was about 76 Quadrillion Btus.
1 Btu = 0.2930 watts.

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fuel demands may be rapidly altered due to one or a combination of changes. Recent factors such as ecological conservation can also effectively block construction of new facilities and lead to a greater delay in supplying the required energy.

SURFACE MINING

Surface mining is a broad term which refers to the removal of the soil and strata over a mineral or fuel deposit and the removal of the deposit itself. The various unit operations in surface mining include:

1. preparing the surface
2. drilling
3. blasting
4. overburden removal
5. loading of the deposit
6. haulage of the mined deposit
7. reclamation

Other operations include the building of drainage systems, waste handling, and topsoil and subsoil storage. Surface mine planning and engineering, therefore, transcends the fields of engineering and physical sciences. Primary considerations include: geology, rock and soil mechanics, excavation and materials handling systems, and environmental aspects.

Surface mining of coal is an answer to many of the underground coal mining problems but in itself creates new problems. Recovery rates with surface mining are much higher than that from underground mining, generally 80 to 90 percent, and coal losses are limited to spillage and losses in transit. Safety problems are not severe in surface mining, but acid drainage from pit and spoil areas must be considered. Land reclamation is an expensive undertaking. A limiting factor with coal surface mining is the depth to which coal can be profitably mined. Currently in most areas the maximum depth of overburden for economic mining is slightly over 60.96 meter (200 ft). Surface mining at greater depths may be feasible in the future, depending upon technology, economics and environmental concerns. The rising trend in coal production in the United States should continue through the rest of the century. The contribution of surface mining to coal production has progressively increased. Today, surface-mined coal accounts for more than 50% of the total coal production (Figure 1). However, the average thickness of coal removed has remained constant, as opposed to an increasing average overburden thickness (Figure 2), a result of major advances in surface mining technology.

Considering the problems which confront the mining and electricity generation industries in their quest to meet the demands for energy, surface mining will be challenged to provide coal at a low cost with minimal disturbance to the environment. With respect to the regulation of surface coal mining operations and to the establishment of performance standards, the passage of the Surface Mining Control and Reclamation Act of 1977 represents a significant effort on the part of the federal government. The U.S.

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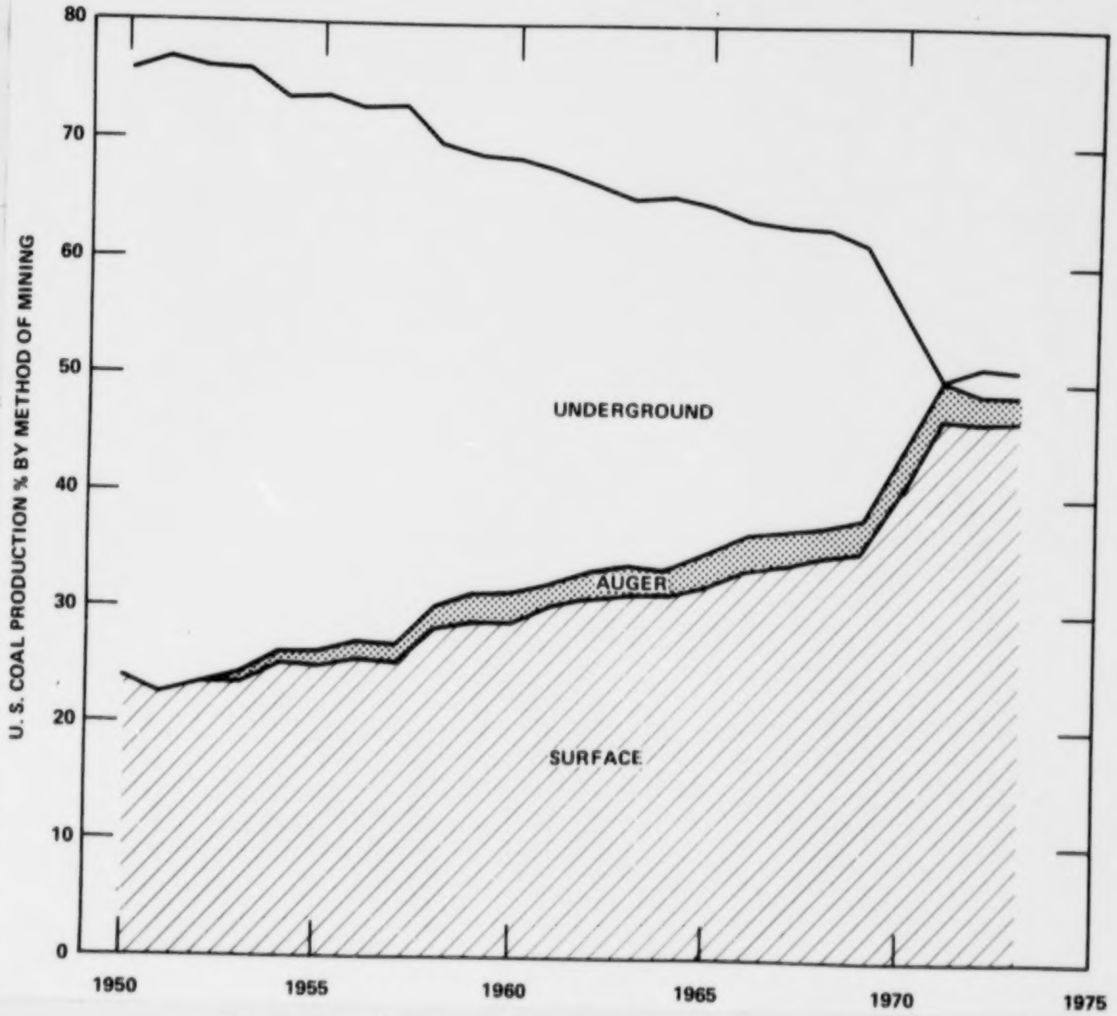

Figure 1. Historical trend of domestic coal production by method of mining (EPRI, 1977).

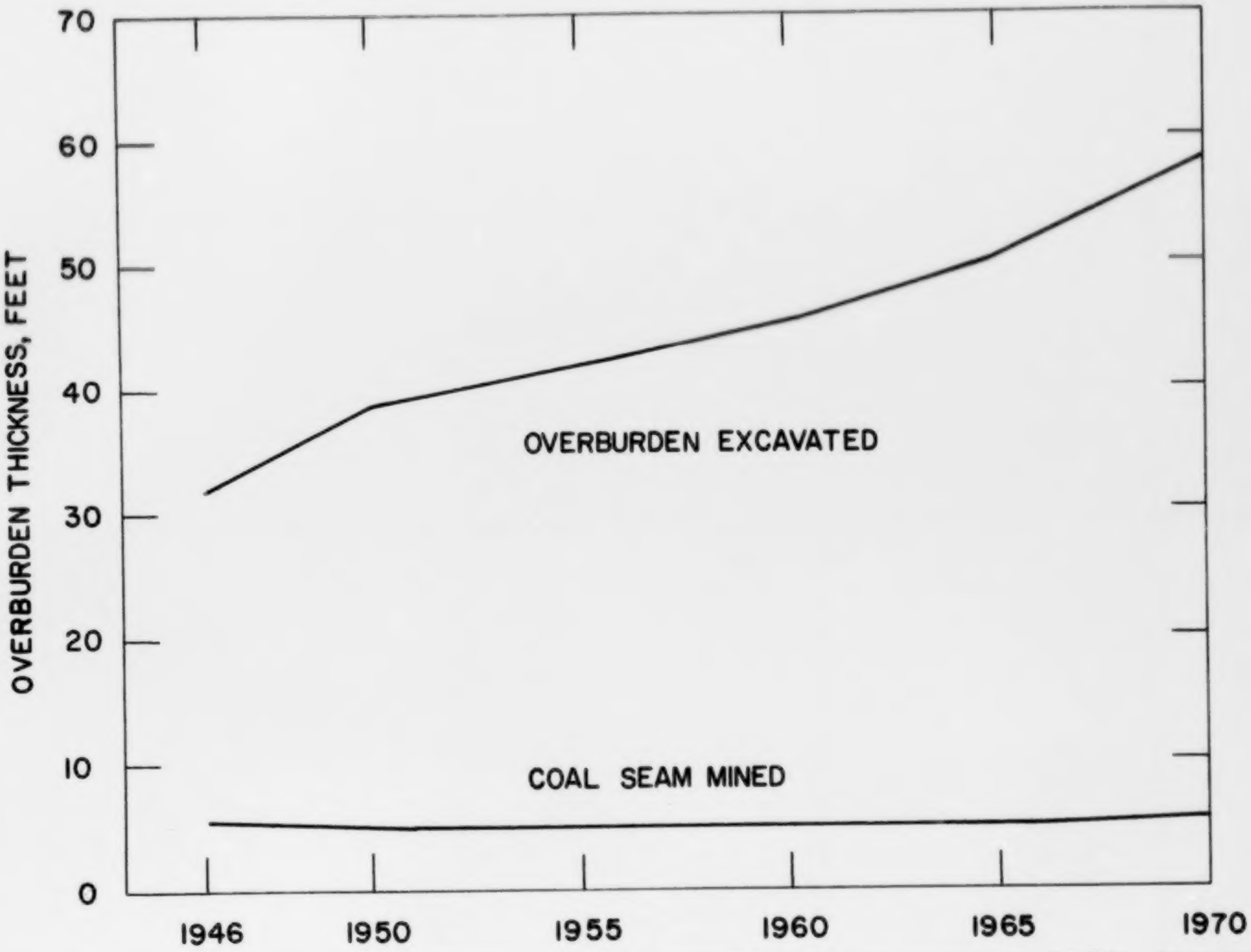

Figure 2. Average thickness of coal seam mined and overburden excavated in the U.S. (EPRI, 1977).

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Congress, according to the Act, finds and declares that "surface mining and reclamation technology are now developed so that effective and reasonable regulation of surface coal mining operations . . . is an appropriate and necessary means to minimize as far as practicable, the adverse social, economic, and environmental effects..." Among the purposes of the Act, important ones are to assure that (1) surface mining operations are not conducted where reclamation required by the Act is not feasible, (2) surface coal mining operations are conducted to protect the environment, and (3) adequate procedures are undertaken to reclaim surface areas as contemporaneously as possible with the surface coal mining operations. While the increasing energy demand promises a good market for coal, increasing federal, state, and local statutes with regard to surface mining make the exploitation of coal deposits a difficult but challenging task. In order to meet this challenge, a need exists to critically study and evaluate surface mining of coal in detail and provide suggestions for solutions to these problems. Attempts to provide solutions to problems require a thorough knowledge of existing methods and criteria. It is the purpose of this report, therefore, to briefly discuss the various aspects of surface mining of coal and lay out important planning and design considerations.

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SECTION 2

EXPLORATION

INTRODUCTION

Exploration includes activities and evaluations necessary to gather data for making decisions on such topics as the desirability for further exploration, feasibility of mining, size, initial flowsheet, and annual output of new extractive operations (Bailey, 1968). However, the main purpose of exploration activity is to facilitate the discovery and acquisition of coal deposits that may be extracted and processed economically, now or in the future. The extent of exploration programs will vary according to company practices and the area under investigation. The program should go to the point of detail where additional study would not provide greater reliability.

The bulk of exploration work is usually assigned to an exploration department staffed with geologists and engineers. Other staff departments that perform considerable work in exploration are legal, public relations, and accounting. The highly simplified, primary role of the exploration department is to determine the existence, location, quantity and quality of a coal deposit; assess the possible favorable and unfavorable factors of mining; and begin the acquisition program for mineral and surface rights and ownership of the required lands. A legal staff should determine prevailing federal and local laws and practices and provide backup for any required negotiations.

TYPES OF EXPLORATION

There are two general types of exploration approaches, regional and detailed. Regional exploration is a reconnaisance activity and is intended to determine if and where a deposit may exist. Detailed exploration is performed on a specific target area where a deposit is known to exist and delineation and verification are desired.

Areas subjected to exploration procedures may be broadly classified into lands where coal seams are known to exist and virgin areas. Coal exploration programs may be defined as follows:

1. Geologic prospecting with minimum drilling. This type of exploration is common in Appalachian regions where outcrops are abundant and geologic formations are well known.

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2. Random drilling. This method is applied to previously explored regions with high geologic predictability.
3. Pattern drilling and statistical approaches. A program of this type is necessary in virgin areas or in areas of high geological variability.

The importance of a thorough exploration program to a coal mining company cannot be over-emphasized, particularly when viewed with regard to surface mine engineering and design. In the eastern coal mining areas with which this study is concerned, the presence or absence of coal is usually known. However, coal mining companies still require much information about overburden and coal characteristics, especially with respect to environmental considerations and present laws and regulations.

The initial step in all exploration programs is a literature search to obtain all public information and previous exploration data concerning a property. Publications and maps may be available from the U.S. Geological Survey, the U.S. Forest Service, state geological services, and other federal and state agencies, and county offices of clerk, assessor, and surveyor. Extensive drill hole information may be available in areas where oil and gas exploration has occurred. Records of mining companies that have previously operated in an area are helpful if they can be obtained. Volume 3 of this research effort discusses the geologic structure of the coal basins and provides a useful summary of the reported literature.

Maps are important tools in all phases of a mining operation. Maps should at least show geology, political boundaries, and surface contours. Special maps may yield aerial views and paleotopographic and paleoenvironmental information. Good maps and mapping techniques will provide a means for planning and accomplishing exploration, development, reclamation, day-to-day operations, and equipment moves. Calculation of material volumes, location of physical elements, and determination of mining conditions are expedited by the use of maps. Maps provide a method for recording data so that they can be organized and analyzed for ready reference.

Maps suitable for use as base maps are available from a variety of sources. A wide range of map scales and types are available. Various laws require certain scales for reclamation and mining plans but this may vary from state to state. The most commonly used scales are 1" = 500', 1" = 400', and 1" = 100'. U.S. Geological Survey 7-1/2' or 15' topographical maps can be easily enlarged to 1" = 1000' and are the type most commonly used for exploration base maps. Other map scales are available but are suitable for more specific uses. These will be discussed in reference to the appropriate topic.

Aerial photography and mapping methods (photogrammetry) are increasing in usefulness in the surface coal mining industry. For some companies, they are indispensable. They serve as a base for mining plans, vegetation, and geologic and structural identification. Infrared aerial photography is useful for historical/archaeological investigations. Photogrammetric methods are relatively easy and cheap, can be adjusted to any scale, and are highly accurate in any terrain.

Aerial photography done at an altitude designed to produce maps of the scale 1" = 100' is quite popular. This method will provide for control that is accurate to one meter in 5000 meters horizontally and one-half of a contour interval vertically. The usual contour interval is 1.52 meter (5 ft) in normal terrain and 3.04 meter (10 ft) in rough terrain. An advantage of this type of mapping is that it may show drainage configuration, roads, buildings, lakes, streams, timber, power lines, railroads, and fences or other features that might be missed by a ground survey. Table 2 provides a list of data that should be included or at least considered in a mapping program.

GEOLOGICAL ANALYSES

Geological analyses are employed by most companies and yield crucial information with regard to pre- and post-depositional forces and the effects of migrating chemical constituents from the overburden on coal. Physical features that can be identified include faults, folds, rolls, dikes, coal water courses, and the presence of concretions and sulfur. The occurrence of seam pinch-outs, local variations in properties and thickness of the coal, and the presence of erosion and channel sands can also be determined by a geologic analysis in conjunction with drilling information. The probability of soil siltation or dust problems can be assessed. Physical properties of coal and overburden that can be determined by physical inspection are specific gravity, hardness, color, luster, size stability, weathering, grindability, friability, fracture, cleat and cleavage. Geologic analysis will also identify chemical and geochemical factors such as toxic or neutralizing elements in overburden and coal.

EXPLORATION PROCEDURES

Good and rapid exploration programs will facilitate pit design and equipment selection whereby costly lead times for start-up can be reduced. Analyses for mining method, preparation processes, and market conditions will be completed sooner so that more profitable decisions can be made. General procedures for exploration are as follows:

1. Property for a new surface mine.

Step a. Acquiring and consolidating sufficient reserves to justify capital expenditures for a viable mine plan.

Step b. Gathering reliable information about coal beds and mining conditions. Exploration, especially drilling, must be performed with proper engineering control to ensure accuracy of the information required for evaluation. The correctness of reserve calculations, along with other factors such as size of property, topography, depth to coal, highwall stability, and thickness of the seam, will have a high influence on equipment selection. Occasionally, the target area may have surface features which preclude mining, thereby reducing available reserves. The scope of the study will vary from property to property and according to the apriori and actual information already available.

TABLE 2. FEATURES TO BE IDENTIFIED ON AN EXPLORATION MAP

Topography

- Natural features
- Surface contours
- Drainage configurations
- Streams - continuous and intermittent
- Lakes
- Rock croppings
- Coal outcroppings
- Natural cavern openings

Man-made features

- Fences
- Wells
- Reservoirs
- Railroads
- Power lines
- Oil and gas lines
- Water and sewage lines
- Highways, roads and streets
- Buildings
- Old mining areas - surface and underground openings
- Location of drill holes - collar elevation, coal elevation, thickness of coal, depth of hole

Geographic features

- Private and political property boundaries
- Property ownership - surface and mineral
- Forests and vegetation
- General land use
- Annual precipitation
- Centers of population - labor supply and housing
- Unusual concentrations of plant or animal life
- Climate altitude

Geological features

- Washouts
- Burn lines
- Faults and fractures
- Major geological formation
- Groundwater - courses, water table classification, relation to coal

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TABLE 2. (continued)

Geological features

- Topsoil - thickness and condition
- Overburden - special characteristics, thickness, chemical features
- Coal - depth, thickness, physical and chemical properties
- Dip of beds
- Presence of methane concentrations
- Partings

Hydrology

- Properties distribution, and circulation of surface and groundwaters

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2. Property with an existing surface mine. This type of exploration is usually intended to expand reserves or further delineate known reserves. The analysis required may be different from the one above and involve merely the accumulation and organization of additional data.

Before actual field exploration can begin, several requirements must be satisfied. The two important items are the landowner's permission for entry and a prospecting permit.

Formal land acquisition procedures are usually initiated after exploration begins. Acquisition procedures, however, should be a continuing part of exploration procedures. Drilling programs are conducted under options to lease or purchase a particular property. Leases are negotiated with the federal government, state governments, and private landowners. Purchase agreements are negotiated with private landowners.

It is important that a mining company control the surface as well as the minerals during the period of mining activity. Complex land ownership patterns in eastern coal mining areas make this objective difficult to attain. Another problem resulting from the complex ownership patterns is the difficulty to put together a block of land containing reserves large enough to be mined economically. This problem is most common east of the Mississippi River.

After a thorough review of the available information from public and private sources, additional information may be obtained only from additional studies. Regional exploration may provide opportunities to use geophysical methods in conjunction with a drilling program. Gravity surveys and seismic reflection and refraction techniques may be helpful in determining the location and geometry of marker beds, faults, and basement rock. Large area exploration may also employ satellite or airborne photography, infrared imagery, and side-looking radar in connection with a drilling program. Side-looking radar is particularly helpful in the Appalachians where the surface growth is relatively dense.

The most common type of field exploration is a drilling program. The purpose of a drilling program is to provide a comprehensive data base covering the location of the coal and the overburden. Therefore, drilling must be planned to yield maximum information. Drill hole patterns will vary according to the terrain, geological formations, the presence of coal outcroppings, and the type of exploration program.

When very large areas are being studied, hole spacings will vary greatly and will probably not be in any set pattern. When the program is narrowed to a regional study, a grid pattern is most common. Detailed exploration is performed in areas where coal is known to exist, and therefore, closer spaced drill patterns are required.

One of the main advantages of a drilling program is that it can provide physical samples of the coal and the overburden for chemical and physical analyses. This is accomplished by coring. In most programs, a percentage

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of the holes are cored for overburden and coal. Usually, this ranges from 10 to 25 percent. Holes should be cored from the surface, through the coal, and then below it in order to maximize information about 'spoil' quality.

Air coring and wireline recovery allow for increased recovery up to 100 percent in favorable conditions. Photographing cores as they come out of the hole can provide greater reliability of data. Gamma ray and conductivity logs should be used. The coal core recovery should closely approach 100 percent. If it does not, the analytical information obtained should be considered suspect.

Prospecting along outcrops may also be accomplished with dozer cuts at regular intervals. This gives the attitude of the coal and will help to determine how much outcrop coal (degraded or oxidized coal) must be left as a barrier and the nature of the overburden with respect to machine operation.

Whatever method of exploration is used, the data from drill holes or outcrop openings must be large enough to present an accurate picture of the deposit and conditions. All data should be plotted on maps as soon as it is obtained. Detailed exploration results should be plotted on large scale maps whose scales may vary from 1" = 100' to 1" = 2000'. The scale of 1" = 100' is the most popular. These maps indicate the depth to the coal and the thickness of the coal seam, isopachs (contour lines representing uniform thickness) for coal and overburden, and stratigraphic sections showing a representative profile of the overburden. There are too many instances where not enough information was gathered and investment decisions were made incorrectly.

SUMMARY

In summary, the details of exploration programs for locating and defining surface mineral coal deposits vary widely, depending on the prior availability of data and information about the area. The major steps that are involved, and some of the techniques that can be used are identified in Table 3. Basically, the major steps require a search of available material regarding the nature and occurrence of the coal, a regional exploration program using as many of the airborne geophysical methods as is practical or possible, and then site specific ground work using drilling and bulk sampling techniques.

While it may be possible to bypass some of the steps in some areas, it is becoming increasingly clear that exploration programs today must provide mine planners with information not only on the geology and hydrology of the coal deposits but also on factors such as land use, historical and archaeological features, and vegetation and wildlife. Since geological information is basic to many mine planning processes, and exploration is one of the first steps in acquiring this information, it is important to choose the right personnel for geotechnical investigations, identify the information needs and importance and their relationships to mine planning, and use suitable data acquisition systems or techniques. The use or the limitations of the collected data for mine planning purposes must also be known. Tables 4

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through 8 are taken from a paper by Ellison and Thurman (1976) and can serve as a guide in this area. Finally, as shown in Figure 3, exploration is a sequential process where after each stage a decision must be made to either continue or abandon the exploration program.

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TABLE 3. STAGES OF EXPLORATION FOR COAL (BAILEY, 1968)

Exploration Stage	Purpose	Method/Source of Data
Stage 1 Regional Appraisal	o-Availability for Mineral Extraction	Federal, State, Local, Zoning Laws, Land Use Plans
	o-Identification of known problem areas (i) water, (ii) historical/cultural features, (iii) endangered fauna/flora, etc.	Previous history, records, government and private studies
	o-Geologic compilation	U.S. Geological Survey State Geological Survey
	o-Market availability and needs	Intelligence data, growth potential, plans of utilities
Stage 2 Reconnaissance of the Region/ or Target Area	o-Infrastructure Development	Aerial photography Geophysical survey Infrared photography Pilot-hole drilling
	F-Field check on geologic compilation	
	F-Identification of anomalies	
	F-Watershed identification	
	F-Regional land use plans	
	F-Detailed mapping of coal outcrops/coal quality, quantity	
	F-Overburden quality and thickness	
	F-Site location for plants and facilities	
	F-Waste disposal areas	
	F-Wildlife and vegetation	

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TABLE 3. (continued)

Exploration Stage	Purpose	Method/Source of Data
	F&L-Geochemical analysis of streams/sediments	
	F-Historical and cultural features	
	o-Analysis to define more closely probable extraction problems	
	o-Analysis to identify major environmental problems	
Stage 3 Detailed Target Area Investigation	F-Detailed drilling, core drilling of coal	Core drilling Geophysical surveys Aerial photo
	F-Logging (down the hole, geophysical)	
	L-Physical, Chemical and Washability of coal	
	F-Overburden depth, stratigraphic sequence	
	L-Physical, chemical and toxic characteristics of overburden	
	o-Ore reserve compilation	
	F-Groundwater table/ permeability/charge potential	
	F-Sites for box-cut	
	o-Preliminary economic evaluation	

o - Office study; F - Field study; L - Laboratory study.

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TABLE 4. AREAS OF TECHNICAL EXPERTISE IN GEOTECHNICAL INVESTIGATION AND DESIGN (ELLISON AND THURMAN, 1976)

PLANNING PHASE	GEOTECHNICAL INPUT	TYPICAL PRINCIPAL[1,2] GEOTECHNICAL INVESTIGATORS	INVESTIGATORS PROVIDING[2] NECESSARY SUPPLEMENTAL DATA
Feasibility Studies	General Site Evaluation	Mining Engineer-Geologist Structural Geologist	Photogeologist Geotechnical Engineer Hydrogeologist Geochemist Petrologist Seismologist
Conceptual Planning			
Preliminary Design	Site-Specific Investigations	Mining Engineer-Geologist Geotechnical Engineer Hydrogeologist	Geophysicist Photogeologist Structural Geologist Geochemist Petrologist
Final Design	Geotechnical Designs	Geotechnical Engineer Hydrogeologist	Mining Engineer-Geologist Geophysicist Structural Geologist Engineering Geologist

(1)Principal Investigators may vary depending upon site conditions.

(2)An Investigator with proper experience and training may provide necessary expertise in more than one area.

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TABLE 5. THE IMPORTANCE OF GEOTECHNICAL FACTORS TO SPECIFIC MINING CONSIDERATIONS (ELLISON AND THURMAN, 1976)

	MINING CONSIDERATIONS / GEOTECHNICAL FACTORS	SURFACE MINING									SURFACE FACILITIES						
		CUT SLOPE STABILITY	FLOOR HEAVE	SPOILS STABILITY	HAUL ROAD DESIGN	HANDLING OF WATER	RIPPABILITY	REHABILITATION	MONITORING SYSTEM DESIGN	PONTANEOUS COMBUSTION	FACILITIES SITING	STRUCTURES	STORAGE PILES	CUT AND FILLS	TRANSPORT SYSTEMS	IMPOUNDMENTS AND SOLID WASTE DISPOSAL	SPONTANEOUS COMBUSTION
REGIONAL GEOLOGY	GEOMORPHOLOGY	2			2	2		2	2		2	2	2	2	2	2	
	STRUCTURAL GEOLOGY	2			2	2	2	2	2		2				2	2	
	STRATIGRAPHY AND LITHOLOGY	2			3	2	2	2	2		2				2	2	
	REGIONAL STRESS PATTERNS	2	2			2								3		3	
LOCAL GEOLOGY	STRUCTURE	1	1	1	2	1	1	1	1	1	1	1	1	1	1	1	
	STRATIGRAPHY AND LITHOLOGY	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
	IN SITU STRESS	1	1	1		2	2		1	1	2	2		2	2	2	
	SURFICAL DEPOSITS	1		1	1	1	2	1	2		1	1	1	1	1	1	1
	NATURE OF EROSION	2	2	2	1	1	1	1	1		1	1	1	1	1	1	2
	MAN-MADE ALTERATIONS	1	1	2	1	1	2	1	1	2	1	1	1	1	1	1	2
GROUNDWATER CONDITION	UNCONFINED AQUIFERS	1	2	1	1	1	1	1	1	2	1	1	1	1	1	1	
	CONFINED AQUIFERS	1	1	2	2	1	2	1	1	2	1	2	2	2	1	1	2
	AQUICLUDES	1	1	2	2	1	2	2	1	2	1	2	2	2	1	1	
	EFFECTS OF FRACTURES	1	1	1	1	1	1	2	1	1	1	1	1	1	1	1	
SURFACE CHARACTERISTICS	TOPOGRAPHY	1		1	1	1	2	1	1	2	1	1	1	1	1	1	
	SURFACE HYDROLOGY	1		1	1	1	1	1	1	1	1	1	1	1	1	1	2
	EROSION	1		1	1	1	2	1	1		1	1	1	1	1	1	
	TRAFFICABILITY	1	2	1	1	1	1	1	1		1	1	1	1	1	1	
	MAN-MADE ALTERATIONS	1	1	1	1	1		1	1	1	1	1	1	1	1	1	1
	EXISTING AND PROBABLE DEVELOPMENT	1	2	1	1	1		1	1		1	1	1	1	1	1	
ROCK AND SOIL PROPERTIES	RESISTANCE TO WEATHERING	1	2	1	1	1	1	1	1	2	1	1	1	1	1	1	2
	LOAD - DEFORMATION BEHAVIOR	1	1	1	1			1	1		1	1	1	1	1	1	
	STRENGTH	1	2	1	1		1	1	1		1	1	1	1	1	1	
	TIME AND ENVIRONMENT DEPENDENCY	2	2	1	1	1	1	1	1	1	1	1	1	1	1	1	1
	EXCAVATION CHARACTERISTICS	1	2		1		1	1	1		1	1		1	1	1	
	TRAFFICABILITY	1	1	1	1	1	1	1	1		1			1	1		
	ERODIBILITY	1		1	1	1	2	1	1		1		1	1	1	1	2

LEGEND

APPLICABILITY / PROBABILITY OF SUCCESS	IN ALL AREAS	OFTEN	FREQUENTLY	OCCASIONALLY	VERY SELDOM
HIGH	1				1
DEPENDS ON SITE CONDITIONS	2	2	2	2	2
LOW			3	3	

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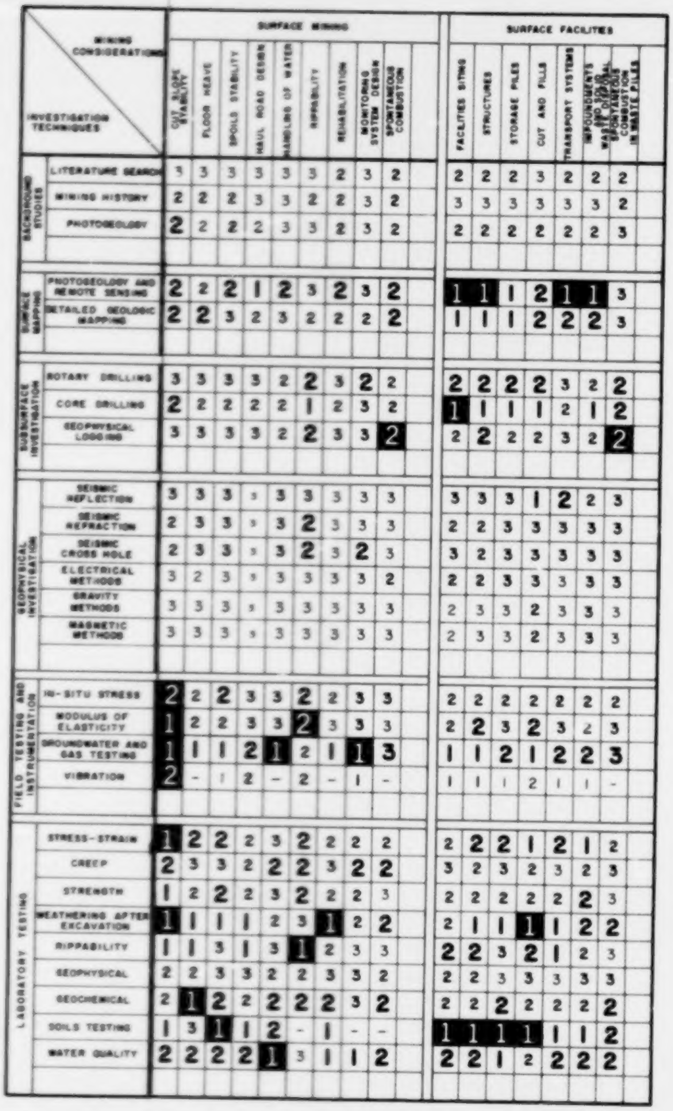
TABLE 6. RELATIONSHIPS OF INVESTIGATION TECHNIQUES TO MINING CONSIDERATIONS (ELLISON AND THURMAN, 1976)



INVESTIGATION TECHNIQUES \ MINING CONSIDERATIONS		SURFACE MINING										SURFACE FACILITES							
		CUT SLOPE STABILITY	FLOOR HEAVE	SPOILS STABILITY	HAUL ROAD DESIGN	HANDLING OF WATER	RIPPABILITY	REHABILITATION	MONITORING SYSTEM DESIGN	SPONTANEOUS COMBUSTION		FACILITIES SITING	STRUCTURES	STORAGE PILES	CUT AND FILLS	TRANSPORT SYSTEMS	IMPOUNDMENTS AND SOLID WASTE DISPOSAL	SPONTANEOUS COMBUSTION IN WASTE PILES	
BACKGROUND STUDIES	LITERATURE SEARCH	3	3	3	3	3	3	2	3	2		2	2	2	3	2	2	2	
	MINING HISTORY	2	2	2	3	3	2	2	3	2		3	3	3	3	3	3	2	
	PHOTOGEOLOGY	2	2	2	2	3	3	2	3	2		2	2	2	2	2	2	3	
SURFACE MAPPING	PHOTOGEOLOGY AND REMOTE SENSING	2	2	2	1	2	3	2	3	2		1	1	1	2	1	1	3	
	DETAILED GEOLOGIC MAPPING	2	2	3	2	3	2	2	2	2		1	1	1	2	2	2	3	
SUBSURFACE INVESTIGATION	ROTARY DRILLING	3	3	3	3	2	2	3	2	2		2	2	2	2	3	2	2	
	CORE DRILLING	2	2	2	2	2	1	2	3	2		1	1	1	1	2	1	2	
	GEOPHYSICAL LOGGING	3	3	3	3	2	2	3	3	2		2	2	2	2	3	2	2	
GEOPHYSICAL INVESTIGATION	SEISMIC REFLECTION	3	3	3	3	3	3	3	3	3		3	3	3	1	2	2	3	
	SEISMIC REFRACTION	2	3	3	3	3	2	3	3	3		2	2	3	3	3	3	3	
	SEISMIC CROSS HOLE	2	3	3	3	3	2	3	2	3		3	2	3	3	3	3	3	
	ELECTRICAL METHODS	3	2	3	3	3	3	3	3	2		2	2	3	3	3	3	3	
	GRAVITY METHODS	3	3	3	3	3	3	3	3	3		2	3	3	2	3	3	3	
	MAGNETIC METHODS	3	3	3	3	3	3	3	3	3		2	3	3	2	3	3	3	
FIELD TESTING AND INSTRUMENTATION	IN-SITU STRESS	2	2	2	3	3	2	2	3	3		2	2	2	2	2	2	2	
	MODULUS OF ELASTICITY	1	2	2	3	3	2	3	3	3		2	2	3	2	3	2	3	
	GROUNDWATER AND GAS TESTING	1	1	1	2	1	2	1	1	3		1	1	2	1	2	2	3	
	VIBRATION	2	-	1	2	-	2	-	1	-		1	1	1	2	1	1	-	
LABORATORY TESTING	STRESS-STRAIN	1	2	2	2	3	2	2	2	2		2	2	2	1	2	1	2	
	CREEP	2	3	3	2	2	2	3	2	2		3	2	3	2	3	2	3	
	STRENGTH	1	2	2	2	3	2	2	2	3		2	2	2	2	2	2	3	
	WEATHERING AFTER EXCAVATION	1	1	1	1	2	3	1	2	2		2	1	1	1	1	2	2	
	RIPPABILITY	1	1	3	1	3	1	2	3	3		2	2	3	2	1	2	3	
	GEOPHYSICAL	2	2	3	3	2	2	3	3	2		2	2	3	3	3	3	3	
	GEOCHEMICAL	2	1	2	2	2	2	2	3	2		2	2	2	2	2	2	2	
	SOILS TESTING	1	3	1	1	2	-	1	-	-		1	1	1	1	1	1	2	
	WATER QUALITY	2	2	2	2	1	3	1	1	2		2	2	1	2	2	2	2	

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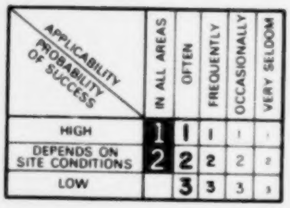

PROBABILITY OF SUCCESS \ APPLICABILITY	IN ALL AREAS	OFTEN	FREQUENTLY	OCCASIONALLY	VERY SELDOM
HIGH	1	1	1	1	1
DEPENDS ON SITE CONDITIONS	2	2	2	2	2
LOW		3	3	3	3

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TABLE 7. ALTERNATE INVESTIGATION TECHNIQUES (ELLISON AND THURMAN, 1976)

	POSSIBLE SUBSTITUTE TECHNIQUE	SURFACE MAPPING			SUBSURFACE INVESTIGATION				GEOPHYSICAL INVESTIGATION							FIELD TESTING AND INSTRUMENTATION					LABORATORY TESTING								
	BASIC INVESTIGATION TECHNIQUE	PHOTOGEOLOGY AND REMOTE SENSING	DETAILED GEOLOGIC MAPPING		ROTARY DRILLING	CORE DRILLING	GEOPHYSICAL LOGGING		REFLECTION	REFRACTION	SEISMIC CROSS HOLE	ELECTRICAL METHODS	GRAVITY METHODS	MAGNETIC METHODS		IN-SITU STRESS	MODULUS OF ELASTICITY	GROUNDWATER MONITORING	VIBRATION		STRESS-STRAIN	CREEP	STRENGTH	WEATHERING AFTER EXCAVATION	RIPPABILITY	GEOPHYSICAL	GEOCHEMICAL	SOILS TESTING	
SURFACE MAPPING	PHOTOGEOLOGY AND REMOTE SENSING		b/2																										
	DETAILED GEOLOGIC MAPPING	b/1																											
SUBSURFACE INVESTIGATION	ROTARY DRILLING					a/2			b/1	b/1	b/1	b/1	b/1	b/1															
	CORE DRILLING				b/1		b/1																						
	GEOPHYSICAL LOGGING					b/2			b/2	b/2	b/2	b/2	b/2	b/2															
GEOPHYSICAL INVESTIGATION	REFLECTION				a/2	a/2	a/1			b/2	b/2	b/2	b/1	b/1															
	REFRACTION				a/2	a/2	a/1				b/2	b/2	b/1	b/1															
	SEISMIC CROSS HOLE																b/1				b/1								
	ELECTRICAL METHODS				a/2	a/2	a/2		b/1	b/1			b/1	b/1															
	GRAVITY METHODS				a/2	a/2	a/2		b/2	b/2	b/2	b/2		b/1															
	MAGNETIC METHODS				a/2	a/2	a/2		b/2	b/2	b/2	b/2	b/2																
FIELD TESTING AND INSTRUMENTATION	IN-SITU STRESS																												
	MODULUS OF ELASTICITY										b/2										b/1								
	GROUNDWATER MONITORING											b/1																	
	VIBRATION								b/2	b/2	b/2																		
LABORATORY TESTING	STRESS-STRAIN					b/2					b/2					b/2	a/2												
	CREEP																												
	STRENGTH					b/2																							
	WEATHERING AFTER EXCAVATION																												
	RIPPABILITY										b/2																		
	GEOPHYSICAL						b/2		b/2	b/2	b/2	b/2	b/2	b/2															
	GEOCHEMICAL											b/2	b/2	b/2															
	SOILS TESTING																												

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a - INDICATES THAT THE ALTERNATE TECHNIQUE CAN PRACTICALLY ALWAYS BE USED IN PLACE OF THE BASIC TECHNIQUE.

b - INDICATES THAT THE ALTERNATE TECHNIQUE CAN SOMETIMES BE USED IN PLACE OF THE BASIC TECHNIQUE.

1 - INDICATES THAT THE ALTERNATE TECHNIQUE SHOULD COST LESS THAN THE BASIC TECHNIQUE.

2 - INDICATES THAT THE ALTERNATE TECHNIQUE USUALLY COSTS MORE THAN THE BASIC TECHNIQUE.

NOTE:

THE USER OF THIS TABLE MUST RECOGNIZE LIMITATIONS IN COMPARING TECHNIQUES FOR GENERALIZED SITUATIONS. THIS TABLE IS A PLANNING GUIDE ONLY.

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TABLE 8. QUALITATIVE EVALUATION OF PRESENT PREDICTABILITY OF MINING CONSIDERATIONS BEHAVIOR (ELLISON AND THRUMAN, 1976)

Mining Considerations / Basis For Predicting Behavior	SURFACE MINING									SURFACE FACILITIES							
	Cut Slope Stability	Floor Heave	Spoils Stability	Haul Road Design	Handling of Water	Rippability	Rehabilitation	Monitoring System Design	Spontaneous Combustion	Facilities Siting	Structures	Storage Piles	Cuts and Fills	Transport Systems	Impoundments and Solid Waste Disposal	Spontaneous Combustion	
Transfer of Technology (1)	**2**	3	**1**	2	**2**	2	**2**	**2**	**2**	**1**	**1**	**1**	**1**	**2**	**1**	**2**	
Field Investigations (1)	3	3	4	3	3	2	**2**	3	3	2	2	2	2	**2**	**1**	4	
Analysis (1)	**2**	**2**	**1**	**1**	**2**	2	**2**	**2**	3	**1**	**1**	**1**	**1**	**2**	**1**	3	
Model Tests (1)	5	5	5					4	5	5	5					5	
Empirical Relations (1)	**2**	4	3		4	**1**	**2**	3	**2**	5	5	3	2	3	3	**2**	
Overall Predictability (2)	2	2	1	1	2	1	2	2	3	1	1	1	1	2	1	3	
Requirement For Monitoring (3)	2	3	5	5	2		1		1	5	5	5	4	5	1	1	

NOTES:

(1) Number designations for predicting behavior vary from:
  1 = Preditive technique is applicable in practically all cases
  5 = Techniques can be applied, but confidence level will be low
(2) Number designations for overall predictability vary from:
  1 = Very predictable in all cases
  5 = Technology and experience for prediction do not exist
(3) Number designations for requirement for monitoring vary from:
  1 = Formal monitoring program required in all cases because of importance, regardless of predictability
  5 = Monitoring can be limited to occasional observations
(4) Darker numbers indicate highest predictive basis for each mining consideration

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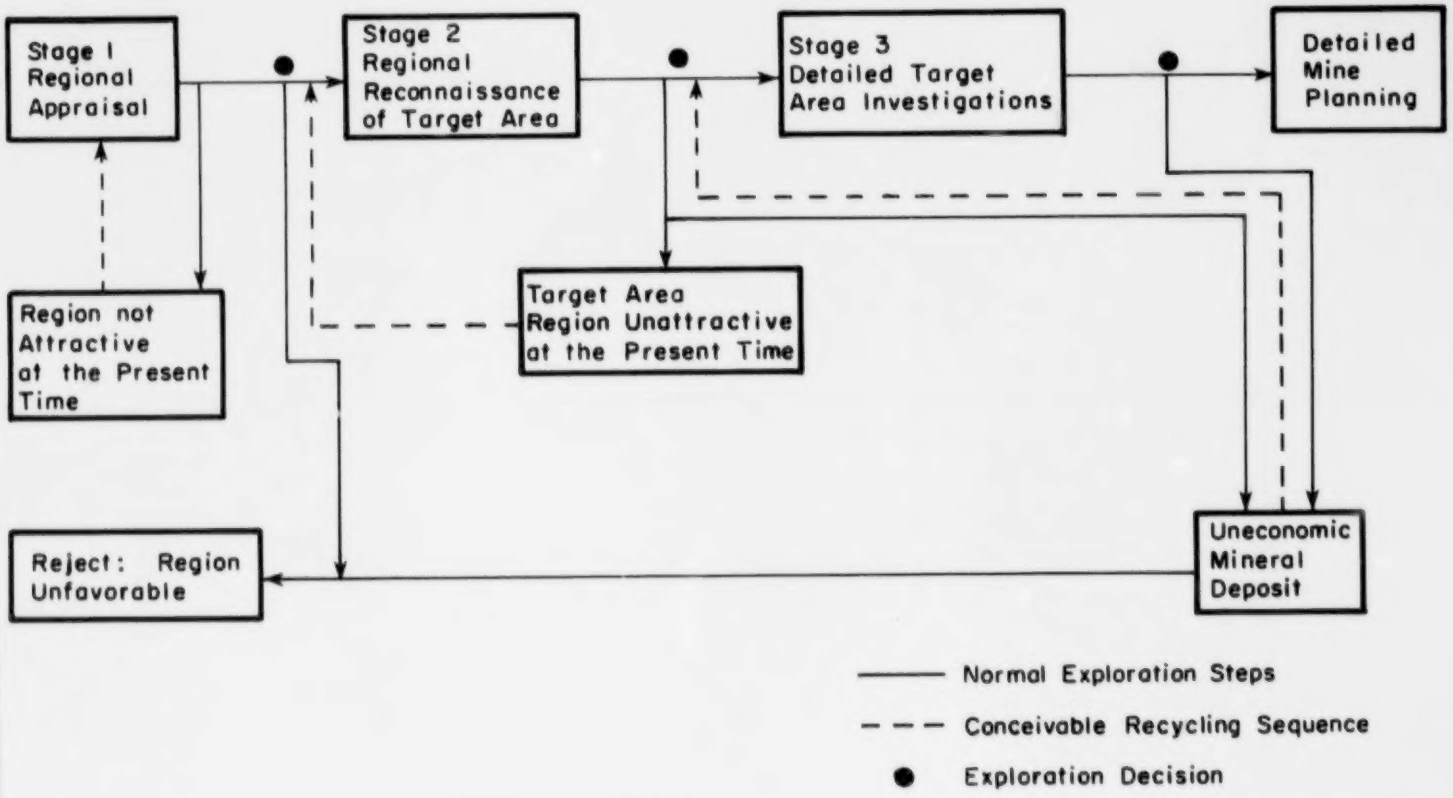
Stage I
Regional
Appraisal
Stage 2
Regional
Reconnaissance
of Target Area
Stage 3
Detailed Target
Area Investigations
Detailed
Mine
Planning
Region not
Attractive
at the Present
Time
Target Area
Region Unattractive
at the Present Time
Reject: Region
Unfavorable
Uneconomic
Mineral
Deposit
Normal Exploration Steps
Conceivable Recycling Sequence
Exploration Decision

Figure 3. Exploration decision flow diagram (Bailey, 1968).

SECTION 3

SURFACE MINING EQUIPMENT

Several types of equipment are available for surface mining, and because of this, a degree of selectivity can be exercised in choosing equipment for a particular job. A brief description of the equipment commonly used in surface mining and their performance characteristics is provided here. Extended descriptions of their physical construction and operational modes are available in many references (Peurifoy, 1970; Pfleider, 1968; Caterpillar Tractor Co., 1975, 1976).

6-1/2"

STRIPPING SHOVEL

The stripping shovel (Figure 4) is a highly productive machine capable of handling all types of material, including large blocky material. However it is restricted to comparatively rigid operating conditions and has a limited ability to spoil. The shovel digs material from its operating base upwards. Factors affecting shovel productivity include low flexibility, low maneuverability, slow tramming rates, bank and bucket slides, and water seepage into the pit. Table 9 gives manufacturer specifications for various stripping shovels, and Figure 5 diagrammatically explains these relationships to the stripping shovel.

DRAGLINE

The dragline is an adaptation of the power crane for earthmoving purposes (Figures 6 and 7). These machines have the ability to dig above and below grade; can function under less rigid operating conditions than shovels; can perform optimally when digging an unconsolidated nonmuddy material; can handle blasted rock; and usually have greater reach than the shovel. Since the dragline operates on the ground surface, the problems of bank slides, water runoffs, and seepage are virtually eliminated. The dragline's ability to "chop down" some fraction of the overburden to dig its own bench (i.e., dig above its operating base) is an advantage for extending its operating range, though this method of operation should be avoided as far as practicable because of significant drops in the machine's production capacity. In any case, due to less precise motions, draglines are only 75% to 80 percent as productive as shovels of equal capacity. Table 10 gives manufacturer specifications for draglines, and Figure 8 shows the relationship between the dimensions.

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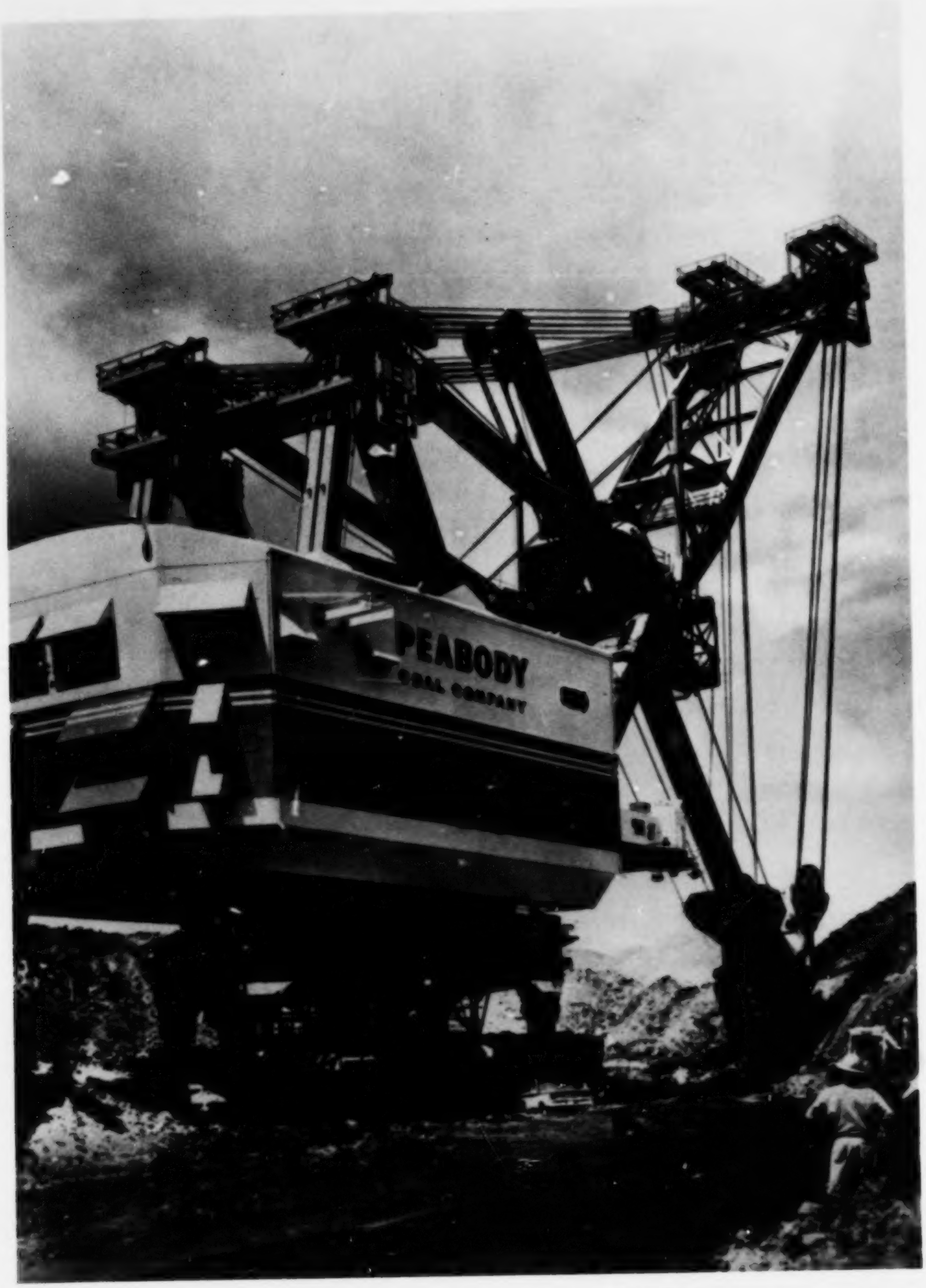
PEABODY
COAL COMPANY

Figure 4. Stripping shovel.

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TABLE 9. STRIPPING SHOVEL SPECIFICATIONS (STEFANKO, ET AL, 1973)

Manufacturer	Model	Boom Angle, Deg. (A)	Dumping Height, Maximum, Ft.-In. (B)	Dumping Radius @ Maximum Height Ft.-In. (C)	Dumping Height Over Spoil, Ft.-In. (D)	Dumping Radius Over Spoil, Ft.-In. (E)	Dumping Height @ Max. Dumping Radius, Ft.-In. (F)	Dumping Radius, Maximum, Ft.-In. (G)	Cutting Height, Maximum Ft.-In. (H)	Cutting Radius @ Maximum Height, Ft.-In. (J)	Cutting Height @ Max. Cutting Radius, Ft.-In. (K)	Cutting Radius, Max., Ft.-In. (L)
BE	550B	45			68-0	100-3	43-0	106-6	92-0			114-0
BE	550B	45			64-0	96-0	43-0	101-6	88-0			110-0
BE	1050B	45			76-0	114-6	48-0	121-0	104-0			131-6
BE	1050B	45			71-0	109-6	47-0	115-0	99-6			126-6
BE	1850B	40	104-0	230-0			83-0	225-0				245-0
BE	1650B	47 1/2	106-6	140-0			69-6	147-6				158-0
M	5761	45	111-9	156-3	106-3	158-0	81-9	162-0	141-3	158-6	88-9	173-0
BE	1650B	47 1/2	96-0	127-6			64-0	135-0				145-0
M	5860	47	122-6	162-6	110-3	167-0	88-3	169-6	152-9	165-0	95-6	181-3
M	5861	46	133-9	180-0	123-6	184-0	92-6	189-0	163-6	182-0	99-6	200-9
BE	1950B	49	134-0	185-0			104-0	193-9				207-6
BE	1850B	47 1/2	101-0	150-6			76-4	157-0				170-0
M	5900	47	143-6	195-6	127-6	200-9	104-6	203-6	176-9	199-0	112-6	216-3
BE	3850B	50	150-0	210-0			108-0	218-0				232-0
BE	1950B	52	117-6	151-6			83-6	163-0				172-0
BE	3850B	50	135-0	192-0			102-0	200-0				215-0
M	5960	45	125-0	181-0	118-0	184-0	95-0	186-6	162-0	185-6	104-0	201-6
M	6360	45	153-0	211-6	133-6	217-9	113-6	219-9	189-6	218-6	122-0	236-9

(continued)

TABLE 9. (continued)

Manufacturer	Model	Radius of Clean-up, Ft.-In. (M)	Length of Crawlers, Ft.-In. (N)	Width of Crawlers, Ft.-In. (O)	Width of Belt, Inches (P)	Length Over Both Crawlers, Ft.-In. (Q)	Width Over Both Crawlers, Ft.-In. (R)	Width and Length, Approximate Ft.-In. (S)	Rear End Radius Maximum, Ft.-In. (T)	Rear End Radius @ Lower Edge of Frame, Ft.-In.(U)
BE	550B	66-0	17-3	9-0	30	47-0	38-9	22-9x48-1 1/2	33-6	
BE	550B	66-0	17-3	9-0	30	47-0	38-9	22-9x48-1 1/2	33-6	
BE	1050B	75-6	21-2 1/8	10-2	36	54-11 1/8	43-11	26-9x52-0	36-0	
BE	1050B	76-0	21-2 1/8	10-2	36	54-11 1/8	43-11	26-9x52-0	36-0	
BE	1850B	125-0	34-0	16-0	66	75-6	57-0	58-0x79-6	57-0	53-0
BE	1650B	97-0	22-7 1/2	13-5	54	58-7 1/2	49-5	36-0x61-8	43-6	
M	5761	95-3	23-4	14-0	60	59-4	50-0	45-0x65-0	45-0	39-0
BE	1650B	84-0	22-7 1/2	13-5	54	58-7 1/2	49-5	36-0x61-8	43-6	
M	5860	99-3	30-0	16-2	66	72-0	58-2	50-0x72-0	50-0	44-0
M	5861	1-5-6	30-0	18-0	72	72-0	60-0	52-0x80-0	56-0	50-0
BE	1950B	113-6	35-0	19-6	78	80-0	68-6	64-0x85-2	62-0	58-0
BE	1850B	109-0	34-0	16-0	66	75-0	57-0	58-0x79-6	57-0	53-6
M	5900	126-3	34-0	21-0	82	78-0	65-0	60-0x86-8	62-0	56-6 1/2
BE	3850B	135-0	41-0	25-8	100	89-0	73-8	70-0x87-0	68-0	63-0
BE	1950B	96-6	35-0	19-6	78	79-0	63-6	64-0x85-2	62-0	58-0
BE	3850B	129-0	41-0	25-8	100	89-0	73-8	70-0x87-0	68-0	63-0
M	5960	124-0	42-0	26-4	104	92-0	76-4	74-0x95-0	67-0	59-0
M	6360	156-9	45-0	30-0	120	103-0	88-0	72-0x104-0	74-0	66-6

(continued)

TABLE 9. (continued)

Manufacturer	Model	Clearance Under Frame, Ft.-In. (V)	Center of Rotation To Boom Foot, Ft.-In. (W)	Ground to Boom Foot, Ft.-In.(X)	Boom Length, Ft.	Dipper Handle Length Including Dipper, Ft.-In.	Dipper Capacity, Cubic Yards
BE	550B	19-0	15-7 1/2	21-7	95	61-6	25
BE	550B	19-0	15-7 1/2	21-7	95	55-6	28
BE	1050B	21-9	17-4	25-1	113	70-0	40
BE	1050B	21-9	17-4	25-1	11w	64-0	45
BE	1850B	29-0	26-1	35-0	235	146-0	65
BE	1650B	24-6	19-3	31-0	145	86-0	70
M	5761	23-6 3/4	20-0	30-6	170	102-6	75
BE	1650B	24-6	19-3	31-0	135	80-6	75
M	5860	27-9	22-0	35-2	180	109-0	80
M	5861	27-9	22-0	36-0	200	118-0	90
BE	1950B	33-1 1/2	29-0	40-0	200	122-0	100
BE	1850B	29-0	26-1	35-0	150	90-6	100
M	5900	33-2	26-0	44-2	200	126-0	105
BE	3850B	38-6	31-6	48-6	210	137-0	125
BE	1950B	33-1 1/2	29-0	40-0	170	102-6	130
BE	3850B	38-6	31-6	48-6	200	125-0	140
M	5960	35-6	28-0	46-6	190	117-0	150
M	6360	39-0	32-0	50-0	215	133-0	180

TABLE 10. DRAGLINE SPECIFICATIONS (STEFANKO, ET AL, 1973)

Manufacturer	Model	Boom Angle, Degrees (A)	Boom Length, Feet	Dumping Radius, Feet (B)	Dumping Height, Ft.-In. (C)	Digging Depth, Ft.-In. (D)	Outside Diameter, Ft.-In. (E)	Width of Shoe, Ft.-In. (F)	Length of Shoe, Ft.-In. (G)
M	151M*	46	80	65	38-0	40-0		3-6	20-9
M	183M*	42	120	100	60-6	40-6		5-6	30-0
M	D/E 7400	30	235	220	104-0	117-6	31-0	6-0	36-6
P	732	30	175	173	64-0	115-0	37-0		
M	7500	30	220	209	84-0	130-0	37-0	6-0	36-6
BE	480W	40	175	154		80-0	36-0	6-0	42-6
BE	Diesel 480W	40	175	154		80-0	36-0	6-0	42-6
M	195M*	43	120	98	63-0	75-0		4-7	31-3
P	736	30	170	171	59-0	115-0	41-0	7-6	35-0
P	740	30	195	197	70-0	120-0	45-0	8-0	40-0
BE	800W	35	195	179		135-0	45-0	7-6	50-0
BE	1260W	30	225	215		130-0	55-0	10-0	56-0
P	752	30	215	221	75-0	135-0	56-0	10-6	49-0
M	7820	30	275	262	112-0	150-0	50-0	9-0	48-0
BE	1300W	34	235	215		150-0	50-0	9-0	54-0
M	7920	30	275	263	104-0	135-0	52-0	10-0	55-0
BE	1350W	38	267	241		150-0	52-0	10-0	55-0
BE	1370W	38	267	241		130-0	58-0	10-0	62-0
M	8000	32 1/2	325	299	136-0	180-0	65-0	13-0	65-0
P	757	30	255	259	95-0	155-0	60-6	11-6	54-6
P	762	38	245	230	113-0	135-0	70-0	12-6	59-0
P	862		275	275	115-0	165-0	66-0	12-6	59-0
M	8050	33	315	290	135-0	200-0	65-0	13-0	65-0
BE	1500W	38	267	241		115-0	63-6	12-0	70-0
M	8200	37	320	282	137-0	150-0	58-0	11-0	55-0
BE	2450W	38	275	249		130-0	63-6	12-0	70-0
M	8400	30 1/2	305	290	120-0	200-0	65-0	12-0	55-2
M	8750	37 1/2	275	246	132-4	126-6	75-0	15-0	70-0
BE	2560W	30	275	274		175-0	65-0	12-0	65-0
BE	2570W	30	285	283		165-0	74-0	14-0	72-0
M	8850	30 1/2	360	340	146-0	180-0	80-0	15-0	70-0
M	8950	34	325	300	137-0	158-0	80-0	16-6	70-0
BE	4250W	38	310	302		185-0	105-0	20-0	130-0

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TABLE 10. (continued)

Manufacturer	Model	Width Over Both Shoes, Ft.-In. (H)	Clearance Radius, Ft.-In. (J)	Boom Foot Radius, Ft.-In. (K)	Clearance Height, Ft.-In. (L)	Boom Foot Height, Ft.-In. (M)	Dumping Clearance, Feet (N)	Boom Point Height, Feet (O)
M	151M*	17-6	18-0	7-5	6-3	8-1		
M	183M*	22-6	23-0	8-3	5-0	10-1 1/2		
M	D/E 7400	44-6	38-0	16-2	4-4	7-3		
P	732		30-10	21-0	5-4	10-0		
M	7500	51-2	39-0	16-2	4-4	7-3		
BE	480W	49-6	38-0	16-8 1/2	4-8 1/2	8-6	33	120
BE	Diesel 480W	49-6	38-0	16-8 1/2	4-8 1/2	8-6	33	120
M	195M*	22-7	26-6	9-5	5-9	10-2		
P	736		41-0	22-6	6-9	10-6		
P	740		48-8	26-0	7-7	11-8		
BE	800W	62-6	50-0	15-3	8-3	14-9	35	127
BE	1260W	77-0	52-0	17-2 1/2	9-0	17-2	42	129
P	752		57-0	32-6	7-8	12-0		
M	7820	71-0	50-0	20-3	7-4	13-6		
BE	1300W	70-0	57-0	17-2 1/2	9-0	17-2	46	148
M	7920	75-0	66-0	21-6	8-7 1/2	12-3 1/2		
BE	1350W	75-3	66-0	25-0	8-6	15-2	48	180
BE	1370W	81-3	66-0	25-0	8-6	15-2	50	180
M	8000	94-0	66-0	21-6	8-8	13-9		
P	757		63-0	36-6	9-2 1/2	15-0		
P	762		75-0	35-0	9-7	16-0		
P	862		63-0	39-0	10-3	17-5 1/2		
M	8050	94-0	66-0	21-6	8-8	13-9		
BE	1500W	89-6	68-0	25-0	8-6	15-2	52	180
M	8200	83-0	68-0	21-6	11-0	14-10		
BE	2450W	90-10	69-0	25-0	8-0	16-0	55	185
M	8400	92-0	66-0	23-3	13-0	19-2		
M	8750	109-0	77-0	24-0	15-10	21-4		
BE	2560W	93-0	80-0	30-0	14-0	16-0	65	153
BE	2570W	104-6	80-0	30-0	14-0	16-0	71	158
M	8850	114-0	84-0	24-0	13-0	21-4		
M	8950	116-6	78-0	26-0	16-0	22-0		
BE	4250W	151-0	105-0	50-0	16-7	24-0	129	214

(continued)

TABLE 10. (continued)

Manufacturer	Model	Point Sheave Pitch Diameter Inches (P)	Bucket Capacity Range, Cubic Yards
M	151M*	38	4-8 1/2
M	183M* D/E	38	5 1/2-10
M	7400	54	9-14
P	732	80	13-19
M	7500	54	10-20
BE	480W Diesel	66	12-18
BE	480W	66	12-18
M	195M*	50	16-18
P	736	80	16-24
P	740	80	21-32
BE	800W	90	16-26
BE	1260W	85 1/4	25-40
P	752	80	36-48
M	7820	82	35-45
BE	1300W	85 1/4	42
M	7920	92	40-60
BE	1350W	120	40-50
BE	1370W	120	52
M	8000	96	45-65
P	757	90	45-64
P	762	110	62-74
P	862	102	55-68
M	8050	96	50-70
BE	1500W	120	65
M	8200	120	60-75
BE	2450W	144	75
M	8400	126	60-80
M	8750	110	80-115
BE	2560W	144	90
BE	2570W	144	115
M	8850	132	110-140
M	8950	132	140-160
BE	4250W	144	220

* Crawler Mounted; BE - Bucyrus-Erie Co.; M - Marion Power Shovel Co.; P - Page Engineering Co.; D/E - Diesel Electric. All other electric unless specified.

NOTE: All dimensions based on largest bucket capacity available. For other bucket sizes consult manufacturers' specifications.

1 ft = 0.3048 m

1 cu yd = 0.7645 cu m

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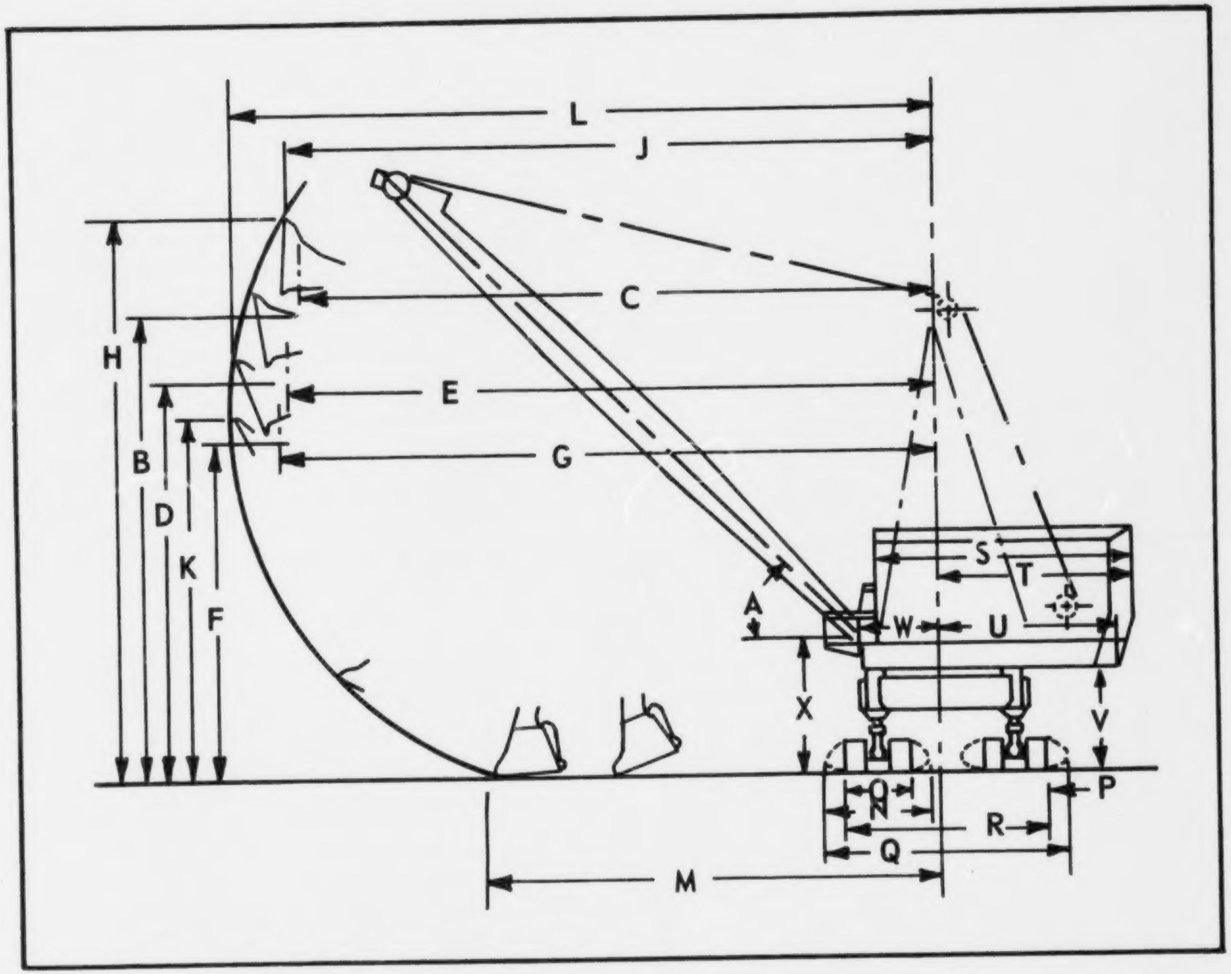
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Figure 5. Stripping shovel working dimensions.

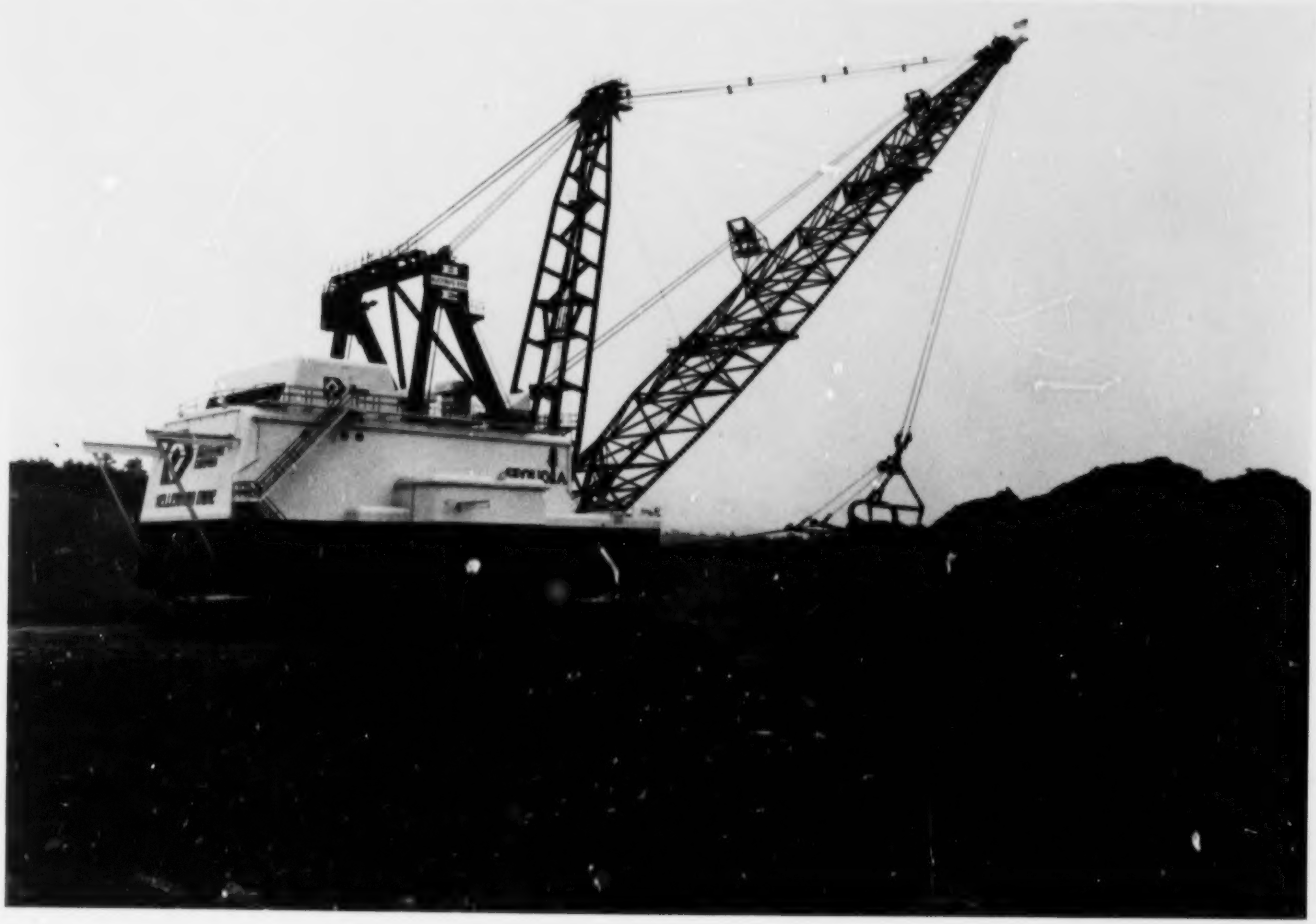

Figure 6. Dragline.

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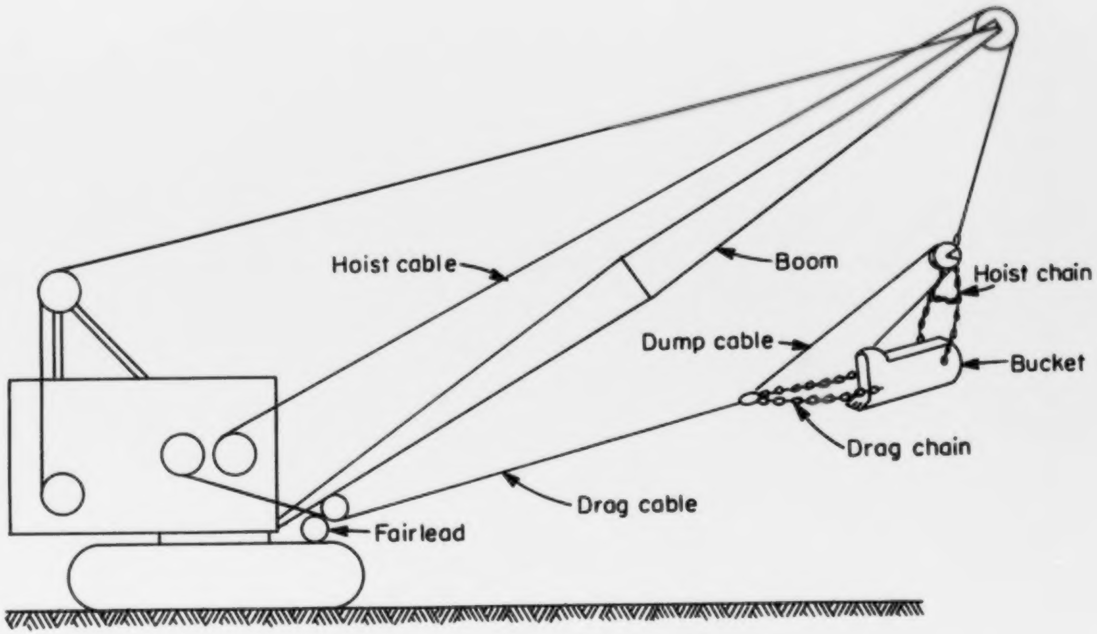
Hoist cable
Boom
Hoist chain
Dump cable
Bucket
Drag chain
Drag cable
Fairlead

Figure 7. Dragline cable configuration.

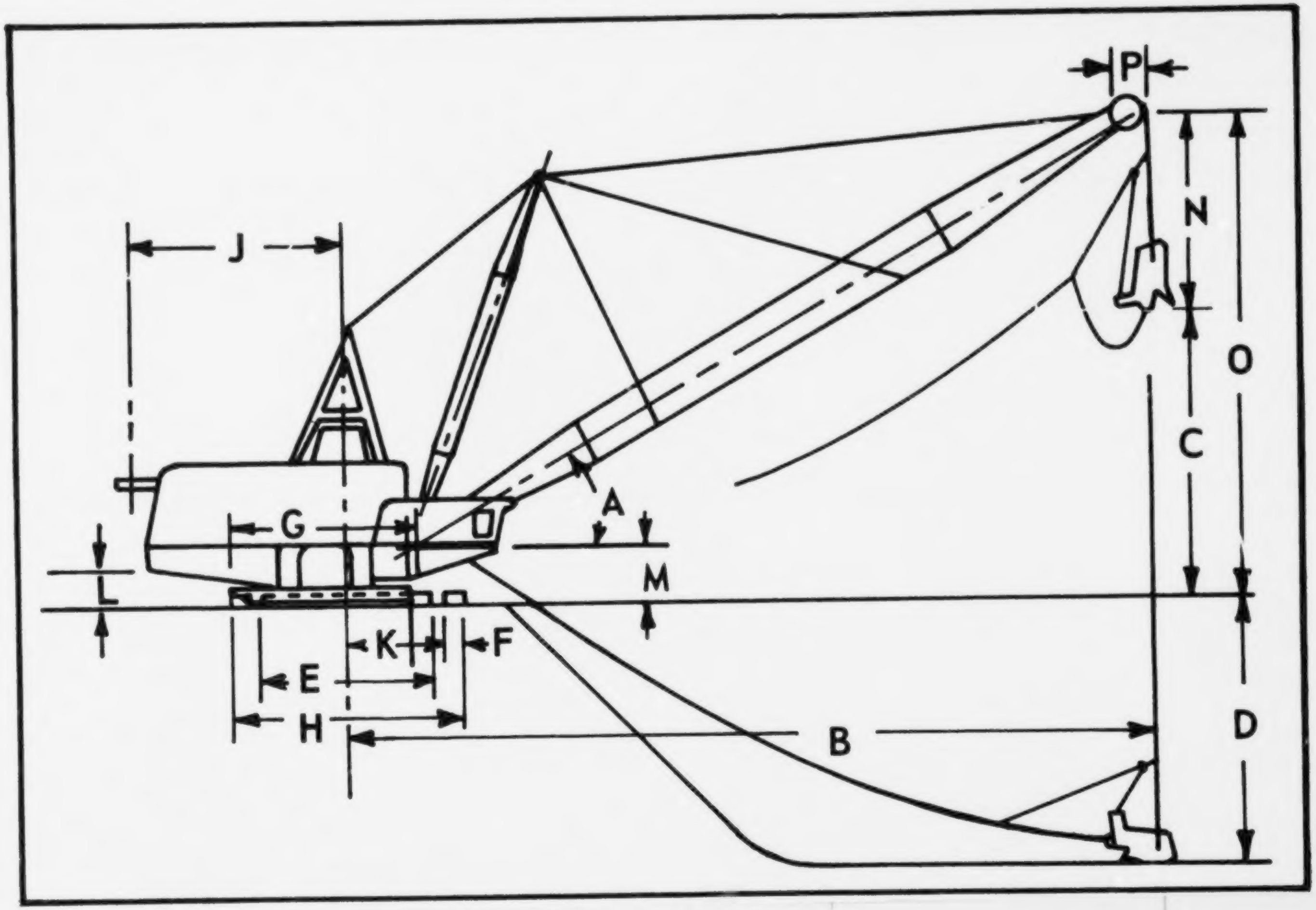

Figure 8. Dragline working dimensions.

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BUCKET-WHEEL EXCAVATOR

The bucket-wheel excavator has its greatest application in unconsolidated overburden where little or no preparation is required. The excavator is adaptable to soft, thin or thick layers of overburden or broken ore, preferably consisting of earth or clay, or sand or soft shales with no hard rock formations present. The bucket-wheel is a continuous mining machine, independent of cyclic operation methods. Most wheel excavators are designed to allow the wheel to excavate below, above, or on the working level, and discharge into belt conveyors which are normally low-cost transportation devices. The long stacker allows a much greater discharge radius while consuming less power per cubic yard of material removed than other equipment (Price, Manula, and Ramani, 1973).

Bucket-wheel excavators for strip mining may be divided into two main types, German and American-Kolbe. The wheel, or digging component, of both types consists of a relatively large diameter wheel with buckets arranged around its circumference and an arrangement to transfer the mined material from these buckets to a conveyor system (Figure 9).

The German type was developed for use in their lignite fields. It is usually crawler mounted, with mountings arranged to give three-point support, and has a relatively low ground-bearing pressure, approximately 103.42 x 10^3 Pascals (15 psi). The ladder (loading) and stacker (discharge) booms swing independently of each other and therefore require counterbalances. Each boom is equipped with a conveyor belt and the transfer of material from the ladder boom to the stacker boom is accomplished through a somewhat complicated chute arrangement. These machines may excavate overburden, lignite, or both overburden and lignite.

Early model German-type bucket-wheel excavators were equipped with both independent swing of the ladder and stacker booms and with "crowd", a feature whereby the ladder assembly is mounted on rollers and a track so that it may be advanced or retracted. As the size of excavators increased, it became uneconomical to retain both independent swing and crowd, so most late model German-type excavators do not have the crowd feature.

Most American-Kolbe-type machines are crawler-mounted with four-point support and have hydraulic cylinders for leveling. Groundbearing pressures are high, approximately 314.76 x 10^3 Pascals (45 psi) because the machines are designed to operate from the coal surface. The ladder and stacker booms do not have independent swing and are not as heavily counterweighted as the German-type machines; since their horizontal relationship does not change, each tends to counterbalance the other. Transfer of materials from the loading to the stacker boom conveyor is simple, since the direction of material flow does not change. These machines usually excavate overburden only and may be used in conjunction with a stripping shovel. Overburden is usually spoiled but may be transferred to an external haulage system.

The Kolbe-type bucket-wheel excavators have crowd. Due to mechanical design restrictions, these excavators do not have independent swing of the

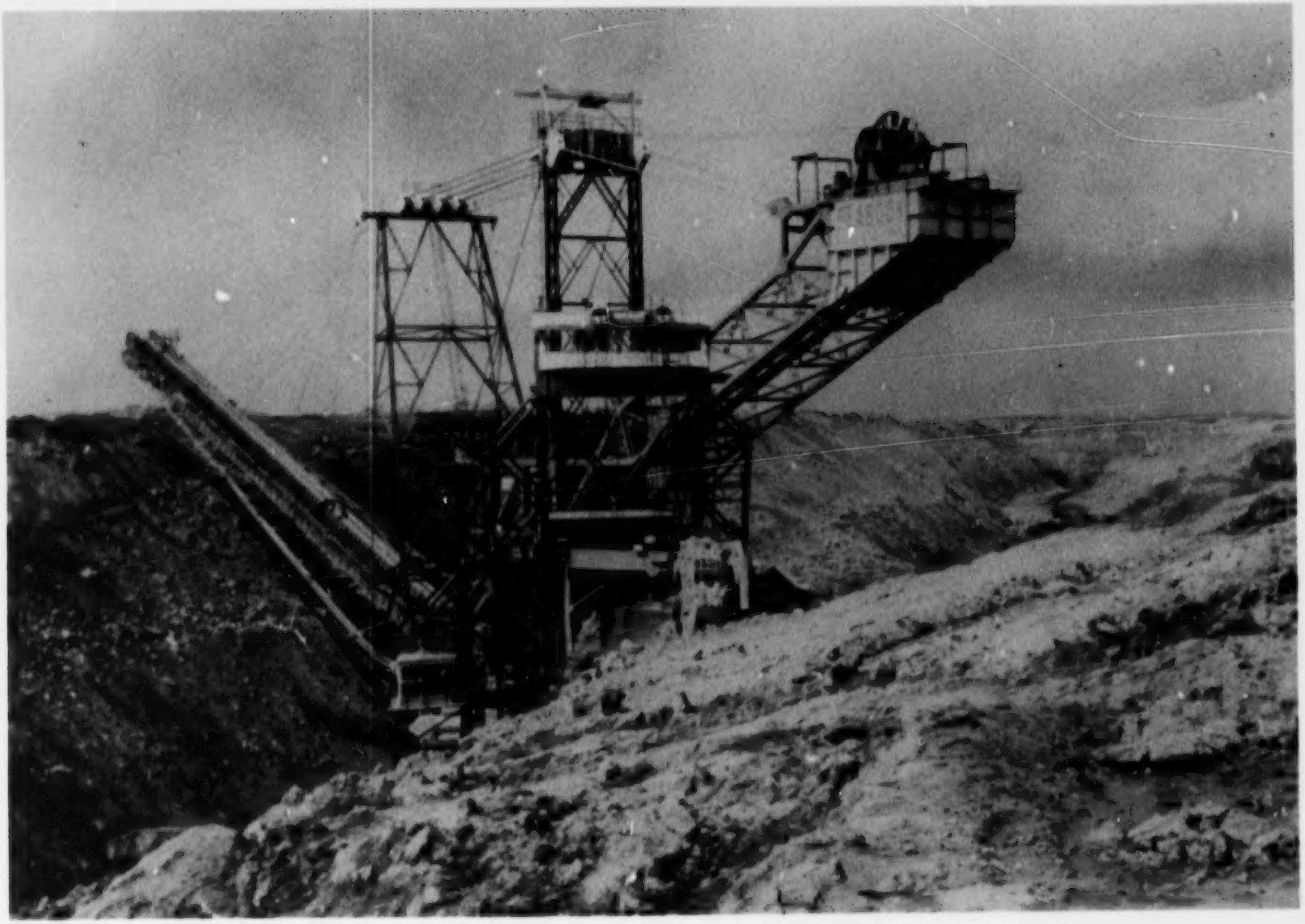

Figure 9. Bucket-wheel excavator.

stacker boom. Several other types of bucket-wheel excavators have been developed in the United States for special applications. These may be either crawler, rubber tire, or rail mounted and have a wide range of capacities.

In contrast to a shovel or dragline, the wheel excavator has a lower instantaneous power demand, no shock loadings, and less weight for greater outputs. Bucket wheels are also an asset for land reclamation, because spoil piles are more regular in size and contour and loose soil can be placed over the hard rock removed from above the coal seam in tandem operations.

Limiting factors for the wheel excavator include less flexibility due to the great bulk of the machine, high maintenance costs, and lower operating efficiency. Other limiting features in bucket-wheel utilization are the considerable working surface preparation required, and the difficulty encountered in handling hard materials or rocks sometimes present in unconsolidated overburden. The loading problems combined with a low break-out force also limit the application of wheel loaders. Safe operation on high benches can also become a problem.

LOADING SHOVEL

The loading shovel in most cases is a scaled-down version of the stripping shovel. The shorter boom of the loading shovel is equipped with either a bottom-dump or a rigidly attached side-dump bucket. The loading shovel, widely used for coal loading in surface mines, is an efficient machine when great mobility is not required (Figure 10). However, it is being replaced at many mines with front-end loaders which can perform better than shovels, especially in contour mining. The loading shovel can load above grade and over the end of a track; can handle coarser materials than the front-end loader; has a longer productive life and better availability; and can load selectively under various mine conditions (Wendertz, 1970).

SCRAPER

Scrapers are of two distinct types, the wheel-tractor scraper (Figure 11) and the track-tractor scraper (Drevdahl, 1961). The main distinction is that the wheel-tractor scraper or rubber-tired scraper, as it is sometimes called, has a speed advantage over the track type, making it more suitable for increased haul distances. The track type, on the other hand, has superior traction in mud; can make its own roads in rough terrain; and can handle larger amounts of material. In surface coal mines the wheel-tractor scraper is more commonly used, although it often requires pushers to assist in loading. When haul distances exceed one mile, scrapers are uneconomical when compared to shovel-truck combinations.

BULLDOZER

A bulldozer is a tractor mounted with a blade which is secured by pivot arms. The blade comes in different types and capacities: U-shaped, "S" or

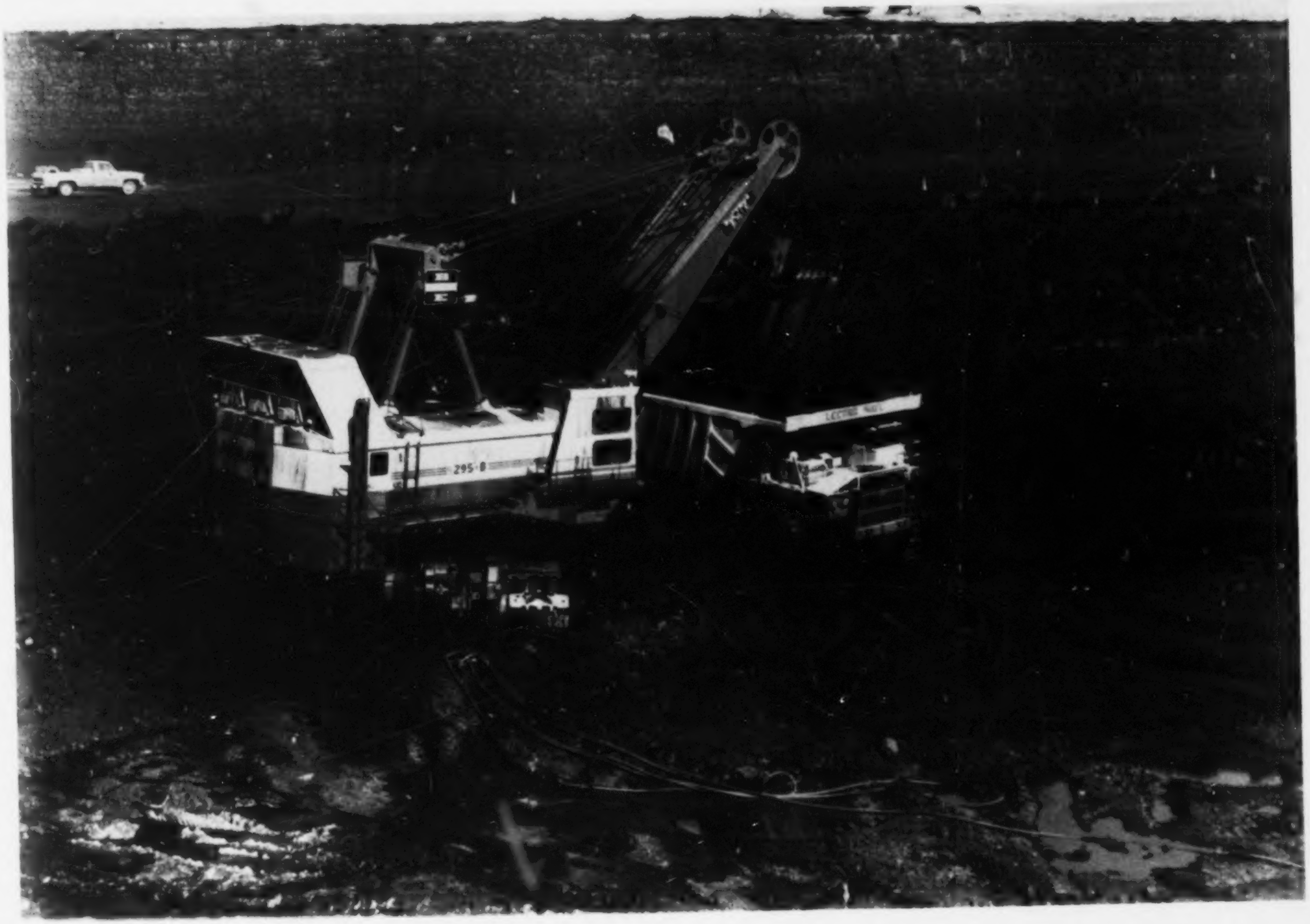

Figure 10. Loading shovel.

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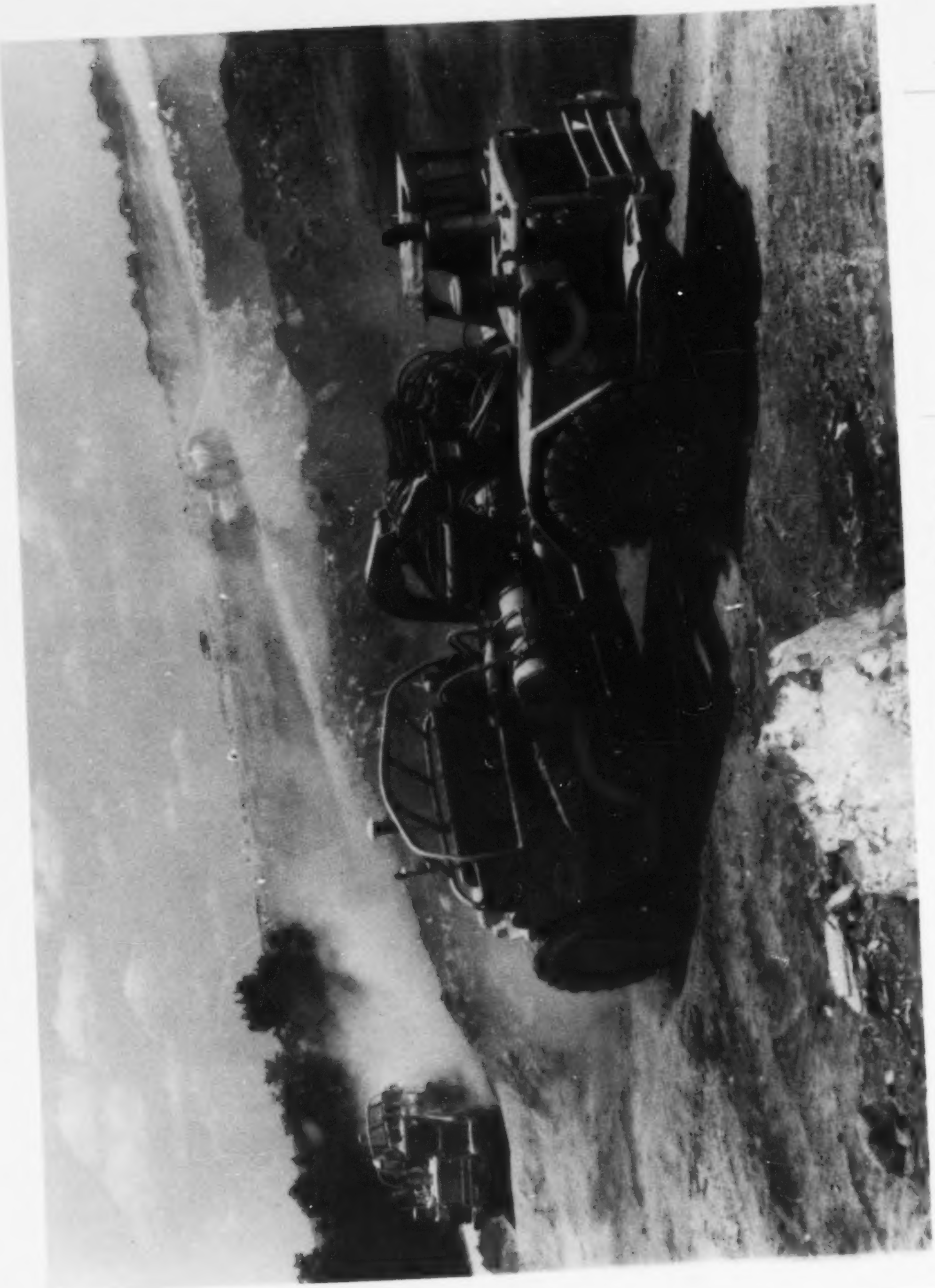

Figure 11. Scraper.

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straight-blade, and "A" or angling-blade. Most surface mine dozer applications utilize special "U" shaped blades for light noncohesive material (Figure 12). The bulldozer is a versatile machine, normally limited to a short operating radius of about 152.4 meter (500 ft). It is applicable for site clearing and preparation; construction and upkeep of access routes and haulroads; cleaning up behind shovels and draglines; boosting trucks from pit; towing disabled equipment; maintaining spoil piles, moving cable skids and stumps; and ripping and disposal. It is also used extensively in stripping and leveling for reclamation.

FRONT-END LOADER

A front-end loader is a wheel tractor with a loading bucket mounted on the front (Figure 13). These machines are efficient for loading coal from seams of small thicknesses (0.304 to 3.048 meter thick (1 ft to 10 ft) where it becomes necessary to gather coal before loading. They have high mobility and can operate under favorable conditions with high productivity. They also have found increased application in contour stripping for load and carry functions over short distances.

HYDRAULIC EXCAVATOR (OR HYDRAULIC SHOVEL)

For mining and other earthmoving operations, hydraulic excavators are adaptations of the traditional backhoe for excavation and loading purposes. Among the advantages claimed for this equipment are the greater break-out force and the variable pitch dipper. Figures 14 and 15 show the use of a hydraulic excavator to remove overburden and load it into a truck.

TRUCKS

In surface mining, off-highway trucks are used primarily for haulage because of their inbuilt ability for continuous uninterrupted operation, while regular highway trucks are used only as service vehicles. There are mainly two types of off-highway trucks, the bottom dump and the rear dump truck. The bottom dump will handle aggregate and unconsolidated material and dumps while in motion. They operate efficiently on grades of less than four percent, and are primarily used for coal haulage in surface coal mines. The rear dump truck comes in various forms: 1) small to medium two-axle trucks, ideal for small pits and development haulage; 2) large two-axle trucks which can handle large loads and negotiate steep grades; and 3) large size three-axle trucks which have identical attributes as the earlier two, but are uneconomical on haul roads due to exhorbitant tire costs (Burton, 1975, 1976).

Off-highway trucks are characterized by engine or drive train type, usually diesel mechanical, diesel electrical, and gasoline internal combustion engines. There are trucks available with either mechanical transmission drive or electric wheel drive. Electrical drive has the advantage of downhill braking the vehicle to eight kmh (five mph) before mechanical or

Figure 12. Bulldozer.

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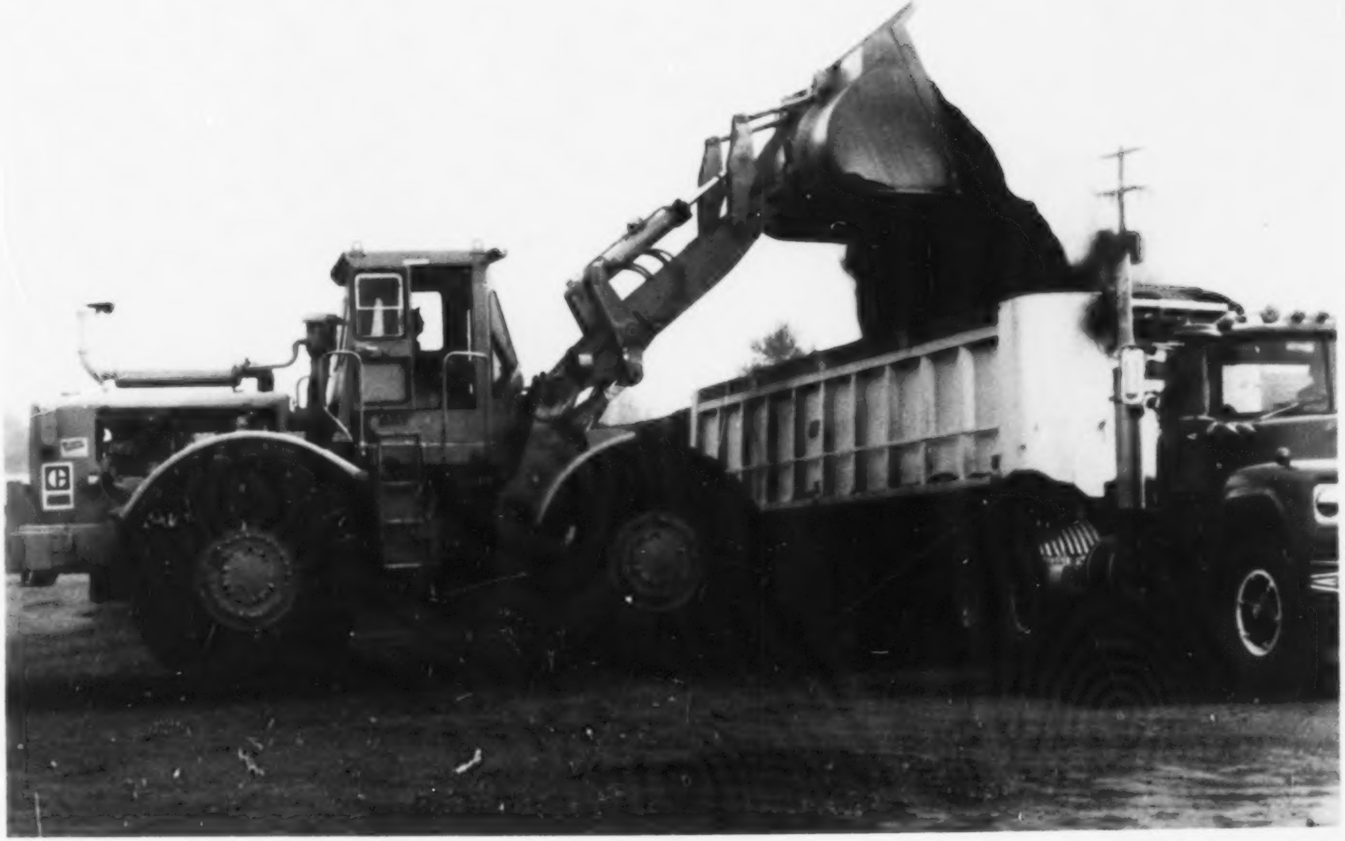

Figure 13. Front-end loader.

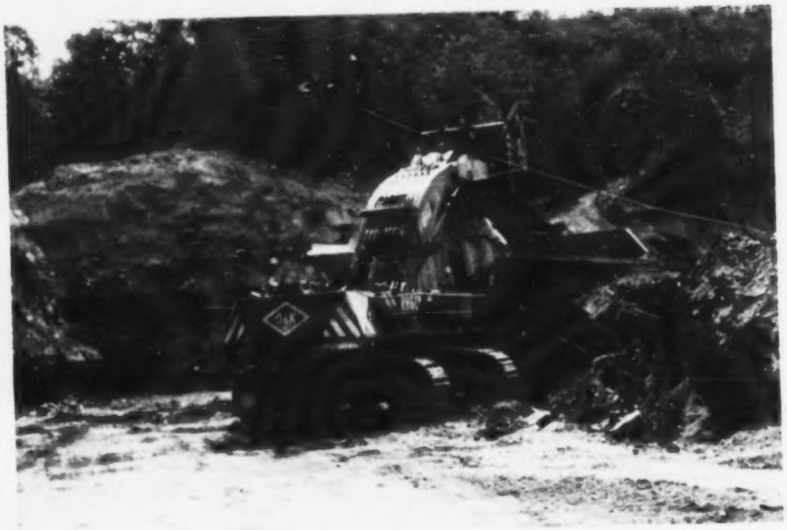

Figure 14. Hydraulic shovel loading into a truck.

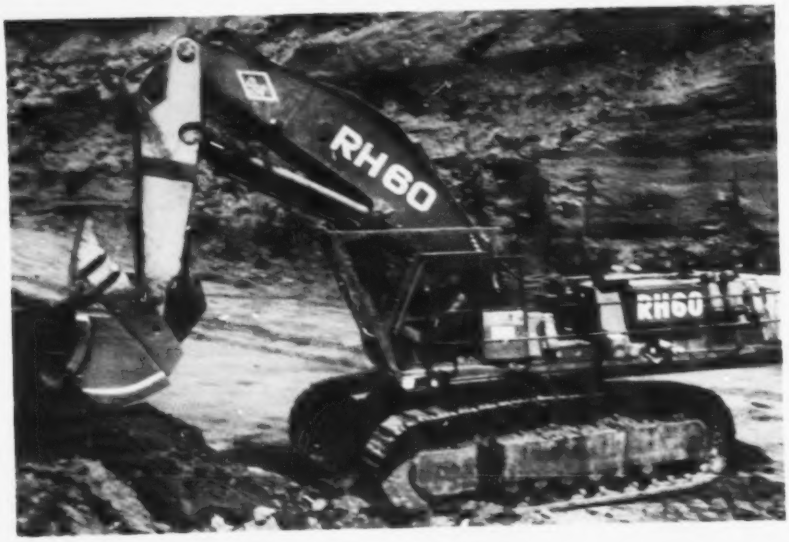

Figure 15. Hydraulic shovel digging into a bench.

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hydraulic brakes are applied for final stop. Off-highway trucks are available in ranges from 26.41 x 10^3kg to 355.61 x 10^3kg (26 tons to 350 tons). Figures 16, 17, and 18 show truck and scraper application zones and the match requirements for trucks and shovels.

EQUIPMENT TRENDS

It is difficult to generalize the equipment preference of operators and manufacturers. Depending on the type of equipment, the trend is toward the selection of medium and large capacities. For instance, the trend in draglines is towards medium capacity equipment with long booms. Long boom draglines have greater operating range, reduce the amount of rehandle, and have improved operating costs.

While no large stripping shovel has been selected in the recent past, a vastly increased use of quarry mine shovels is seen in surface coal operations for both overburden and coal handling. The trend in the loading shovels is towards large capacity machines, i.e., 15.29 cu m to 19.11 cu m (20- to 25-cu yd) shovels. This is a result of the shift to large truck capacities. While several truck manufacturers have 152.4 tonne (150-ton) units in service, in 1976, Wabco introduced its 3200B Haulpak, a 238.76 tonne (235-ton) truck (Felix, 1976). It is evident that the normally small secondary equipment are increasing in size for application with the already stabilizing medium to large size primary stripping equipment.

For secondary equipment such as dozers, front-end loaders, and scrapers, the trend is also toward larger sizes. In contour mining, there is an increase in the use of this equipment for methods which are referred to as "haulback" methods. In these methods, the overburden is transported by scrapers, and front-end loaders over short distances; when the haul is long, usually front-end loaders and trucks are employed. Haulback methods have become more common because of their flexibility in selective spoil placement and in reclamation. While small front-end loaders have not been displaced, large loaders are being employed in coal surface mines. The Michigan 6750, a 18.34 cu m (24-cu yd) loader, has proven useful in eastern surface mines.

Large diameter drill holes with longer drill rods are possible with modern rotary drills equipped with air filtering and ventilating devices. Additional efforts have been directed toward improving the productivity of the current models by increasing the rotary speed and adding automatic controls, thereby allowing operators to fully utilize the drill capabilities.

SUMMARY

This chapter has briefly reviewed the major equipment currently used in surface mining of coal. The need for overburden segregation, topsoil removal, storage and replacement, and selective placement and burial of toxic materials has dictated the use of several pieces of equipment that have loading and carrying capabilities. From a reliability point of view, the largest available machines have not been widely accepted. At

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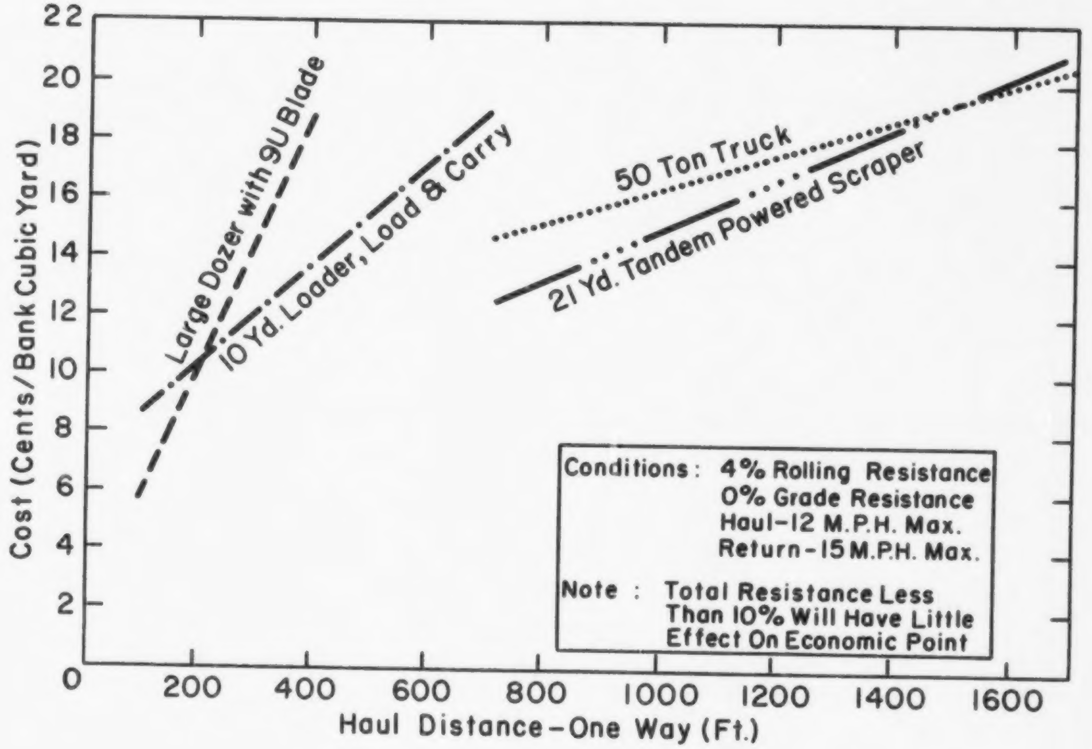

Figure 16. Equipment application zones (Haley, 1974).

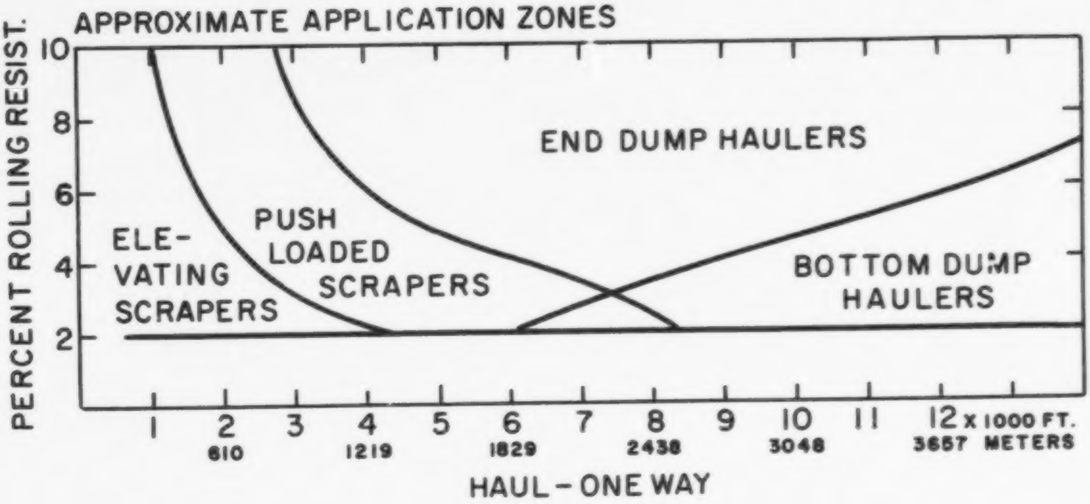

Figure 17. Approximate application zones
(International Harvester, 1975).

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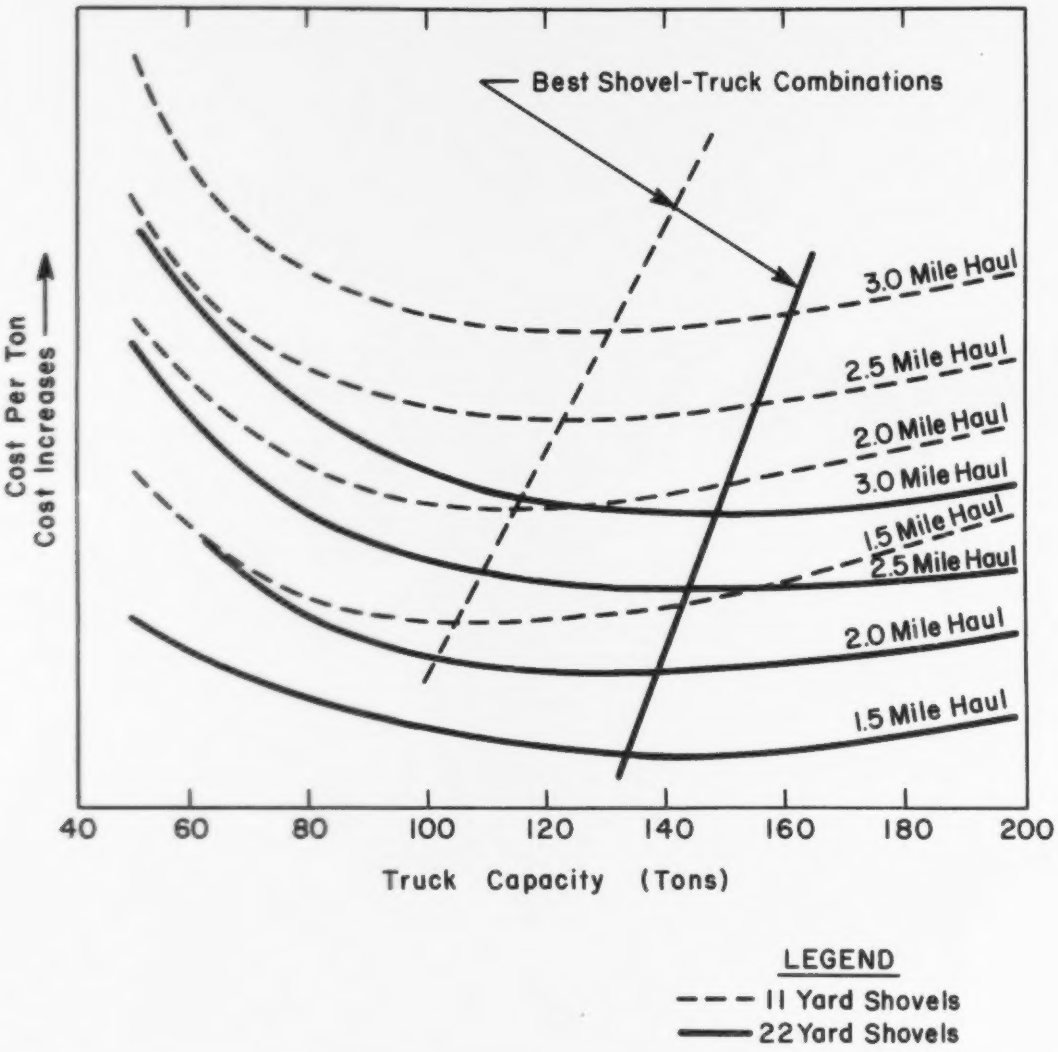


Figure 18. Optimum shovel-truck combinations (Pfleider, 1968).

the present time, there are several research projects funded by the Department of the Interior, Bureau of Mines, for equipment and methods development in surface coal mining. For the near future, draglines, trucks, and shovels are seen as the main overburden removal equipment.

SECTION 4

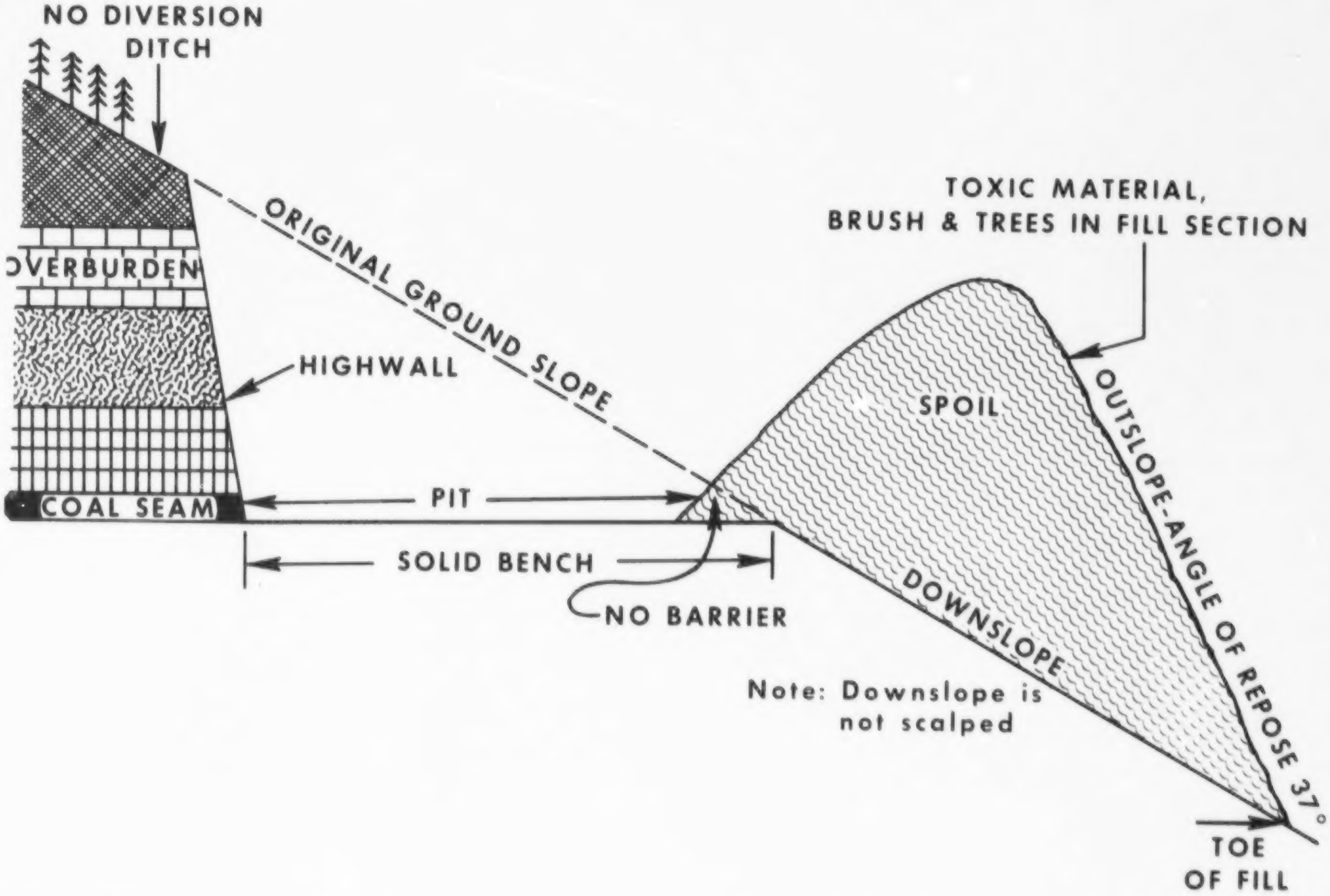
SURFACE MINING METHODS

OVERVIEW

Surface mining of coal is conducted in a relatively simple sequence of operations which includes: 1) preparing the surface, 2) drilling, 3) blasting, 4) overburden removal, 5) loading the deposit, 6) haulage of the mined deposit, and 7) reclamation. Mining techniques for a particular region are largely dictated by geologic and topographic conditions. Even where the techniques are generally comparable, economics of alternative equipment choices and utilization are not easy to generalize. The surface mining techniques can be broadly classified into: 1) contour mining, 2) area mining, 3) open-pit mining, 4) quarry mining, and 5) auger mining. Other methods include dredging and hydraulic mining. Auger mining is usually associated with contour mining where coal near the final highwall is recovered by boring into the seam to a depth of 60.96 meter (200 ft), with cutting heads up to 2.13 meter (seven ft) in diameter. Since quarrying, dredging, and hydraulic mining are not employed for surface mining of coal in the east, they will not be reviewed.

Contour mining is commonly practiced where the coal seams occur in rolling or mountainous terrain. Basically, this method consists of removing the overburden above the coal seam by starting at the outcrop and proceeding along the hillside. After the exposed seam is loaded out, additional cuts into the hillside are made until the economic limit for surface mining is reached.

Contour mining, as practiced in the past, created a bench on the side of the hill (Figure 19). On the inside, it was bordered by the highwall while a high ridge of spoil, the slope of which may cause severe erosion and landslides, was on the outside of the bench. It was not uncommon to punch into the coal seam from the highwall and extract the seam by underground methods (punch mining). It was also common, where there was not enough coal for punch mining, to recover the coal by auger mining (Figure 20). This method of contour mining had come under great criticism. Demands for grading the mined-land to original contour and for the burial of the exposed highwall has made it necessary that the spoil be hauled back into the mined-out pit. Therefore, coal companies are experimenting with several extraction and reclamation techniques that aim at controlling adverse environmental impacts and increasing land values after mining. Several new methods with innovative "spoiling" sequence have been proposed and are being practiced in the steep slopes of eastern United States. In many instances, haulback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ORIGINAL GROUND SLOPE
TOXIC MATERIAL, BRUSH & TREES IN FILL SECTION
OVERBURDEN
HIGHWALL
SPOIL
OUTSLOPE-ANGLE OF REPOSE 37°
COAL SEAM
PIT
SOLID BENCH
NO BARRIER
DOWNSLOPE
Note: Downslope is not scalped
TOE OF FILL

Figure 19. Past practice of conventional contour mining which is no longer an allowable method.

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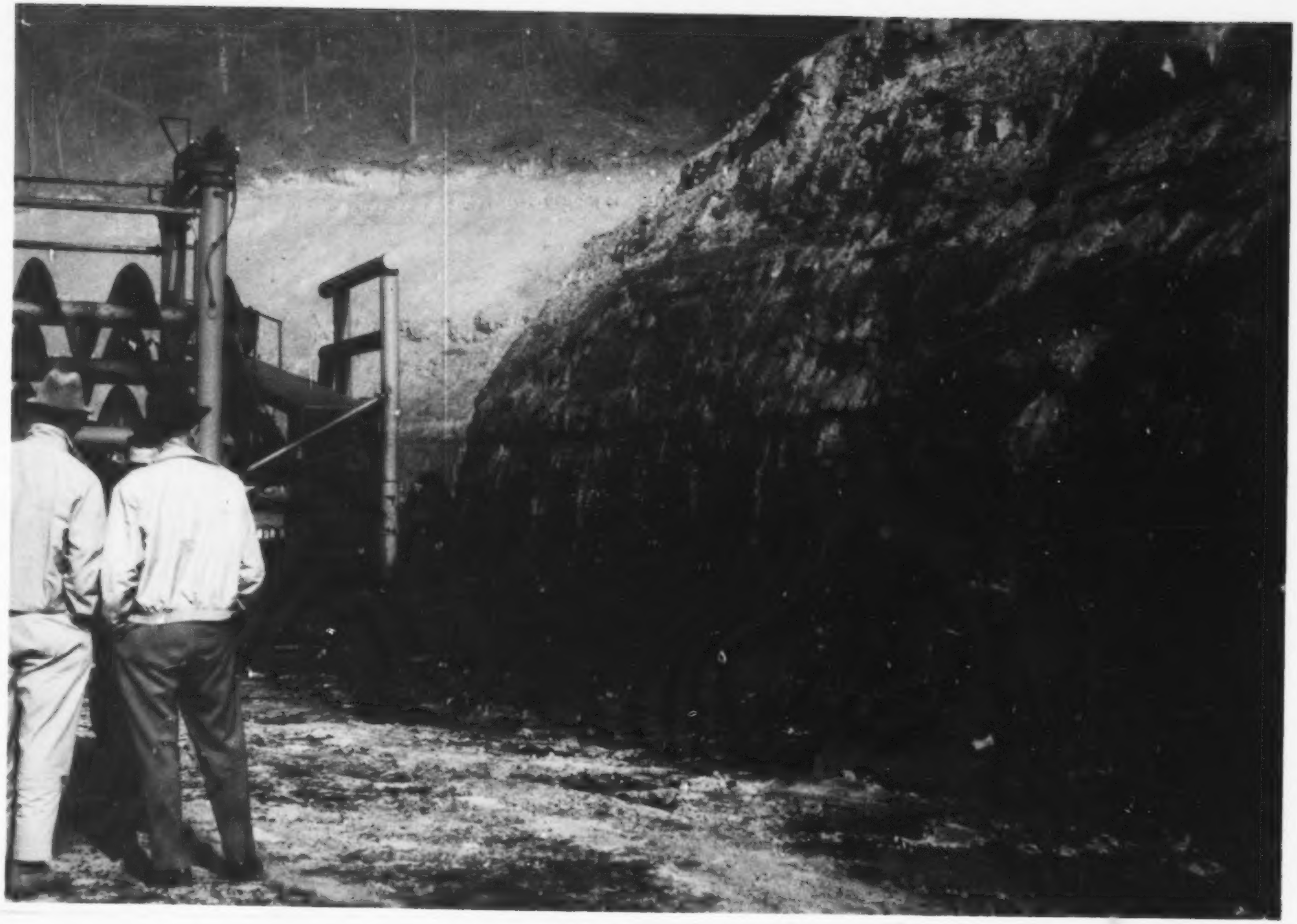

Figure 20. Auger used to drill laterally into a coal bed exposed in contour mining.

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methods have replaced conventional overcasting methods. These methods provide the flexibility to separate top soil, segregate toxic materials, and generally facilitate placement of spoil in a desired sequence. Two such methods are the block-cut and mountain top removal methods.

Area mining, applied where the terrain is flat, commences with a trench or "box cut" made through the overburden, which exposes a portion of the coal seam (Figure 21). This first cut may be extended to the limits of the property in the strike direction. As each succeeding parallel cut is made, the overburden is deposited in the cut previously excavated.

Simple overcasting, explained in Figure 21, is still the most common form of stripping. Shovels and draglines continue to be popular, with draglines increasingly favored over shovels. However, the requirements of topsoil removal, storage, and respreading on the top of graded spoil, and the requirements of textured material in the graded spoil for safe operation of farm machinery have dictated the choice of scrapers or other mobile machinery to handle the upper layers of the soil in a haulback mode of operation.

In open-pit mining of coal, the amount of overburden removed is small when compared with the amount of coal recovered (Figure 22). Thick western coals under shallow cover are often removed by open-pit mining. Open-pit mining may also be used for steeply dipping seams (Figure 23).

Geology of the deposit is a big factor in the selection of the method. All non-stratified deposits and steeply pitching and thick stratified deposits require an outside dump or around the pit transportation of waste. The general method can be classified as open-pit mining. Area mining is applicable to gently pitching, thin deposits. This influence of the geology and the topography are illustrated in Figure 24.

UNIT OPERATIONS

As previously mentioned, surface mining consists of a sequence of seven operations. A discussion of these unit operations follows. This section will also mention the types of equipment that can be employed.

Preparing the surface includes the removal of all vegetative cover in preparation for other mining operations. While this operation is not always necessary, it may be necessary to clear the land of trees or other obstructions. Sometimes, this operation may produce commercial quality timber or may be required for relocation purposes.

It is not easy to define what is topsoil though regulations in several states and federal statutes require that topsoil be removed and stored for later use. Topsoil is generally understood to be the soft layers of soil over which the current vegetation has established its roots. Since this material is soft, elevating scrapers, bulldozers, or front-end loaders

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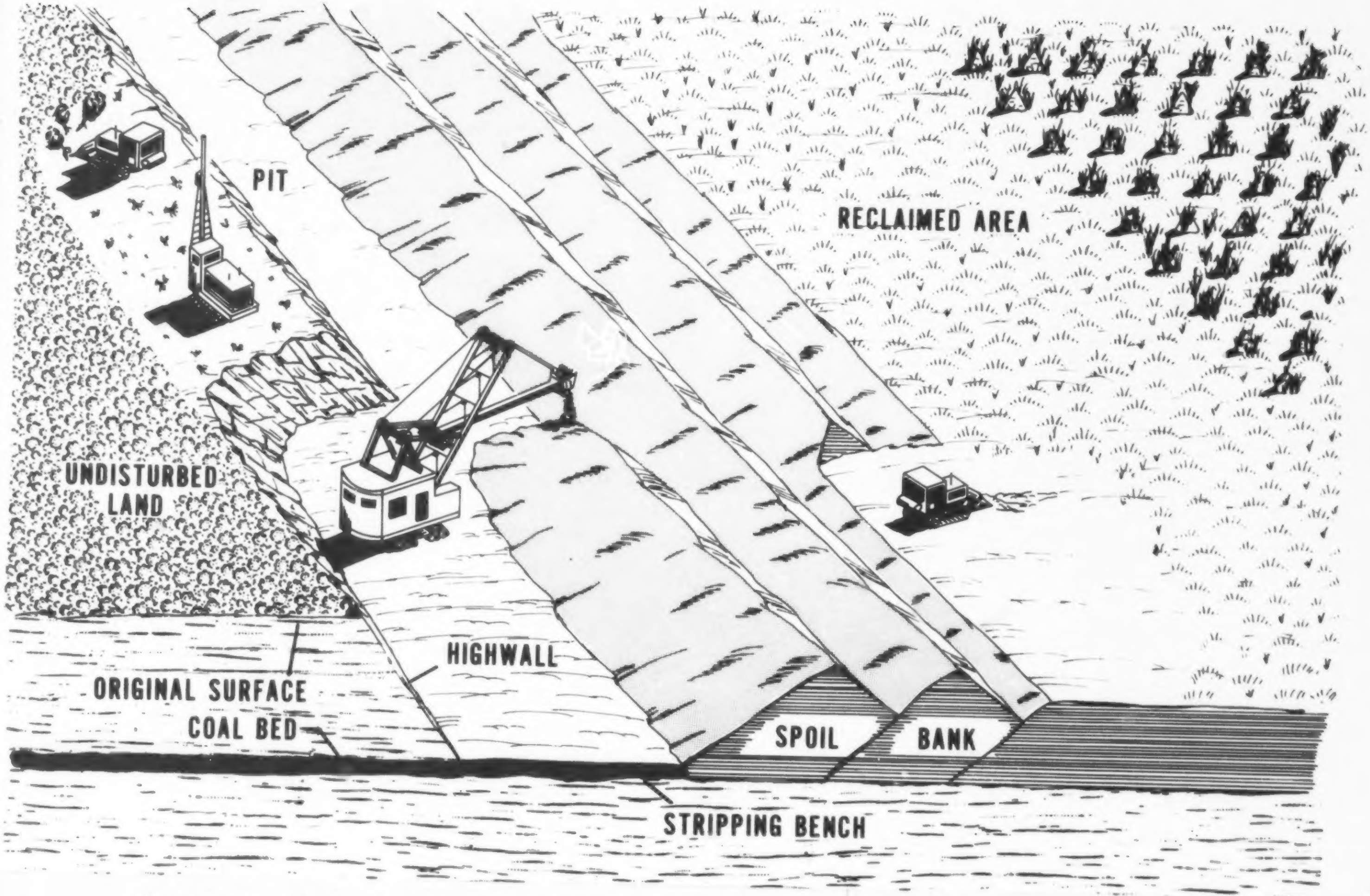

PIT
RECLAIMED AREA
UNDISTURBED LAND
HIGHWALL
ORIGINAL SURFACE
COAL BED
SPOIL
BANK
STRIPPING BENCH

Figure 21. Typical area mining method with stripping shovel (Grim and Hill, 1974).

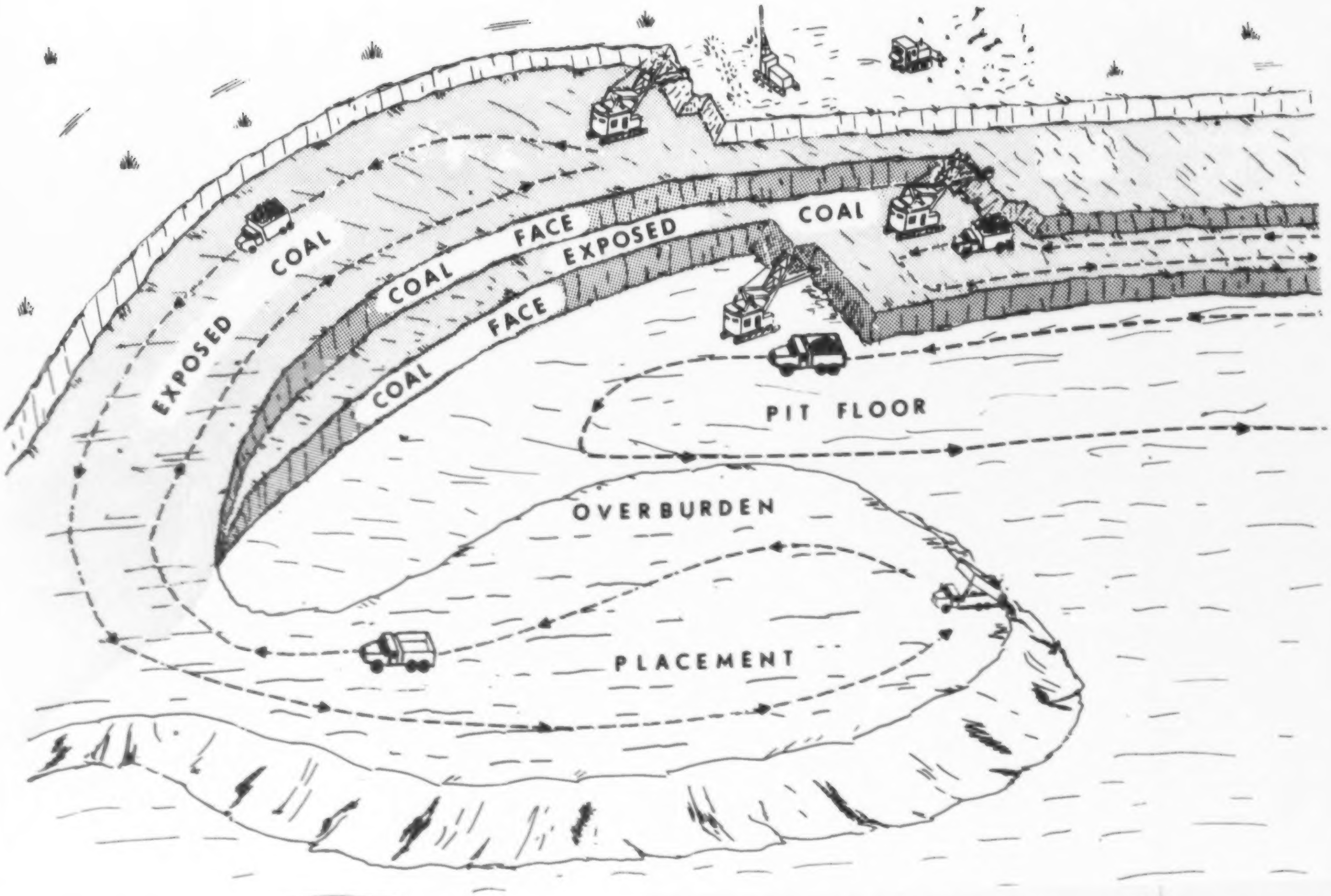

EXPOSED
COAL
COAL
FACE
EXPOSED
COAL
FACE
COAL
PIT FLOOR
OVERBURDEN
PLACEMENT

Figure 22. Typical open-pit mining method (thick seam) (Skelly and Loy, 1975).

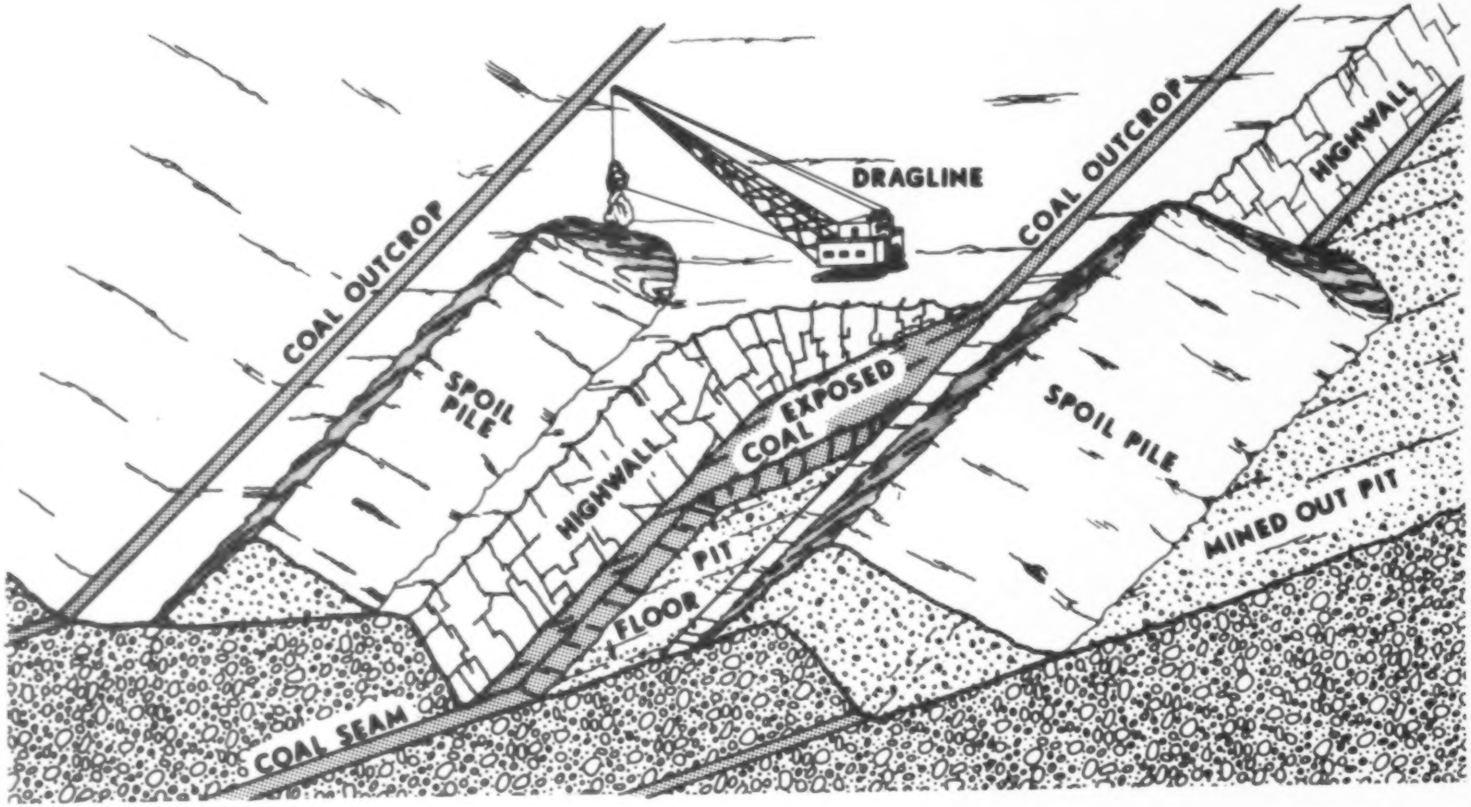
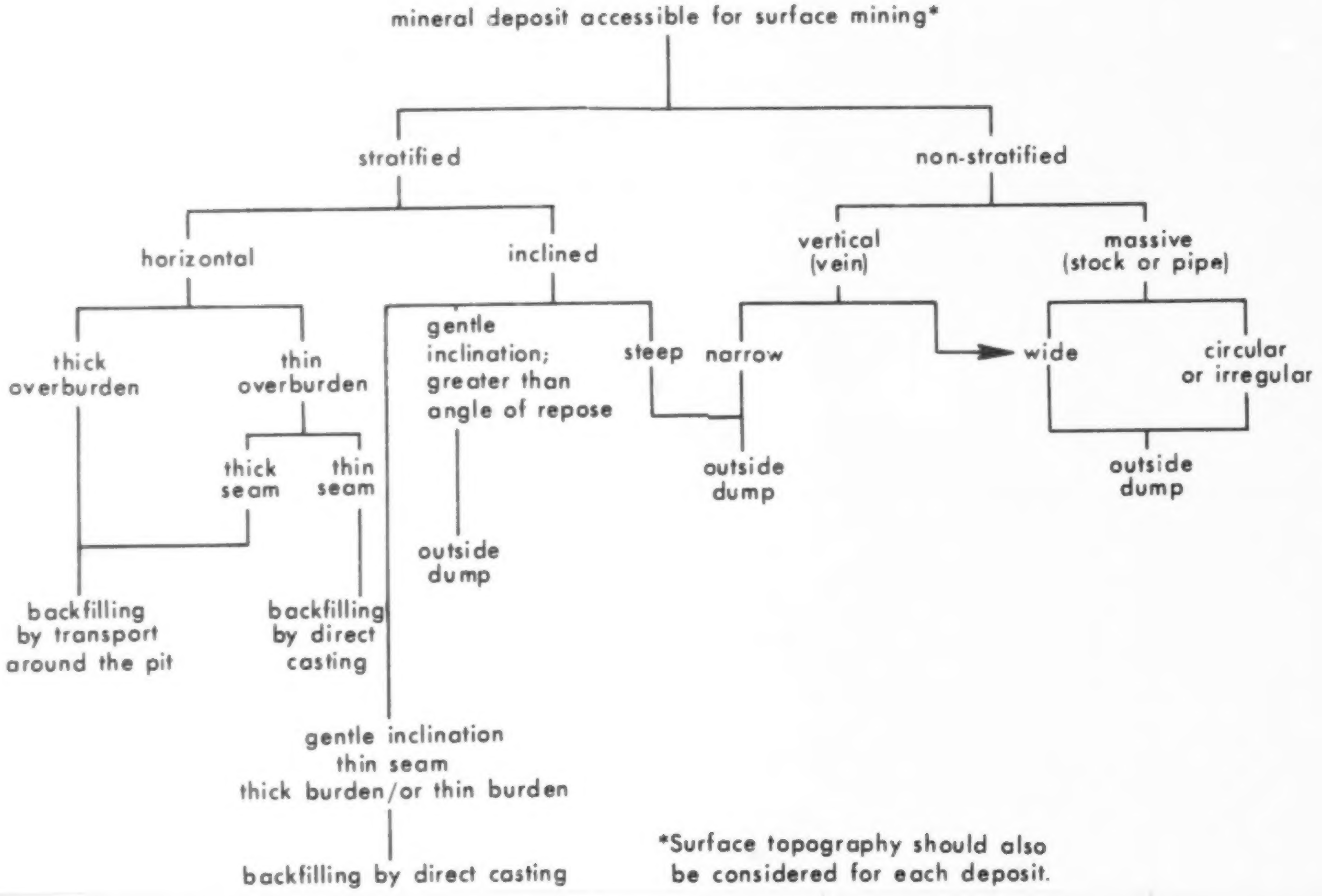

COAL OUTCROP
DRAGLINE
COAL OUTCROP
HIGHWALL
SPOIL PILE
HIGHWALL
EXPOSED COAL
SPOIL PILE
MINED OUT PIT
PIT FLOOR
COAL SEAM

Figure 23. Typical open-pit mining method (dipping seams) (Skelly and Loy, 1975).


mineral deposit accessible for surface mining*
stratified
non-stratified
horizontal
inclined
vertical (vein)
massive (stock or pipe)
thick overburden
thin overburden
gentle inclination; greater than angle of repose
steep
narrow
wide
circular or irregular
thick seam
thin seam
outside dump
outside dump
outside dump
backfilling by transport around the pit
backfilling by direct casting
gentle inclination thin seam thick burden/or thin burden
backfilling by direct casting

*Surface topography should also be considered for each deposit.

Figure 24. Influence of geological factors in surface mine design (Thomas, 1973).

are used for its removal. According to the Permanent Regulatory Program under the Surface Mining Control and Reclamation Act of 1977 (Title 30, Chapter VII, Subchapter A):

> Soil horizons are contrasting layers of soil lying one below the other, parallel or nearly parallel to the land surface. Soil horizons are differentiated on the basis of field characteristics and laboratory data. The three major soil horizons are:
>
> 1. A horizon. The uppermost layer in the soil profile often called the surface soil. It is the part of the soil in which organic matter is most abundant, and where leaching of soluble or suspended particles is the greatest.
> 2. B horizon. The layer immediately beneath the A horizon and often called the subsoil. This middle layer commonly contains more clay, iron, or aluminum than the A or C horizons.
> 3. C horizon. The deepest layer of the soil profile. It consists of loose material or weathered rock that is relatively unaffected by biologic activity.

To allow easier handling by stripping equipment later, the unit operations of drilling and blasting to fracture rock masses may or may not be required. Where the ground cover is hard, it is usually drilled and blasted. Soft strata may often be directly excavated. Some strata can also be sufficiently prepared through use of ripper-bulldozers.

Overburden removal is the most important aspect of the mining system. The equipment and methods used to remove overburden must be carefully chosen to provide the required production at the minimum cost. The stability of the highwall and the spoil is an important consideration in pit design. The equipment used varies from fleets of mobile equipment such as end-loaders to giant draglines, shovels, and bucket-wheel excavators. A factor that has become important is the ability to segregate the soil into stratigraphic layers at desired locations for reclamation and environmental control purposes.

Coal loading is usually done by a loading shovel. Other methods include use of end-loaders, hydraulic excavators, or fine graders. Coal transport is typically by bottom dump trucks.

Reclamation includes backfilling, regrading, surface stabilization, revegetation and restoration operations.

Backfilling is achieved through use of virtually any of the equipment used in overburden handling. However, approaches to segregation of material, specific burial, and layering and compacting can be distinguished as distinct backfilling steps.

Regrading is typically achieved by wheeled or tracked dozers. The amount of regrading depends upon the care with which backfill is done, and on the degree of topsoil restoration and grading, which is either desired or practicable.

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Considerations of surface stabilization include provision for water quality maintenance, compaction and layering, and sealing the spoil layers. It also encompasses amendments and therapeutic actions which revitalize the surface or prepare it for revegetation, such as mulching, fertilizing, liming, stabilizing, disking, harrowing, etc. One phase is usually the spreading of topsoil on the regraded soil. Scrapers are often used to reclaim soil from the stockpile, transport it, and spread it on the spoil. End-loaders may be used with bottom-dump trucks which spread the topsoil "on the run." Seeding is actually the final stage of rehabilitation.

Revegetation and restoration takes place over a longer time period. The growth of plants and establishment of cover is an involved process. Careful attention over several growing seasons may be required. Of course, this step is not necessary where alternative land use, such as a shopping center or a school, is envisioned.

Tables 11 through 17 summarize the unit operations, types of applicable equipment, and ranges of operation, and Figure 25 provides some combinations of equipment that can be employed in surface mining systems for these unit operations.

AREA MINING

Area mining is practiced in the eastern and interior provinces as local conditions permit. It is commonly practiced in Ohio, western Kentucky, Illinois, Kansas, and Oklahoma. In this method, a trench, usually called box cut, is made through the overburden to expose the coal seam. Overburden material (or spoil) thus removed is placed on adjacent unmined land, stabilized and contoured. As the coal is removed from the first cut, a second cut is started immediately adjacent and parallel to the first cut. Overburden removed from this cut is placed in that portion of the first cut from which the coal has been removed. Cut by cut, the mining process proceeds across the property. The final cut leaves an open trench, bordered on one side by the spoil and on the other side by the unbroken column of overburden or highwall. If restoration were not affected, the spoil banks from stripping would resemble a giant washboard or the ridges of a plowed field. The preferable practice in most cases, includes grading the spoil to a level terrain and planting grasses and/or trees.

The practice, depicted in Figure 21, is known as simple overcasting. Simple overcasting, so named because the overburden is cast over the pit to the spoil bank, is the most common form of overburden stripping. The suitability for reclamation, efficiency, and capability of equipment operation, and other considerations, have resulted in the development of alternatives to simple overcasting--passover, crossover, or haul-around techniques. Area mining incorporates a wide range of earth- and rock-moving equipment. The range of stripping applications is presented in Table 18. Table 19 presents some of the combinations which can be practiced in a two-part (multiple-seam) stripping operation. If a three-part stripping operation were to be undertaken, then the third lift would be taken by any method feasible under the second lift.

TABLE 11. EQUIPMENT RATING - TOPSOIL REMOVAL AND REPLACEMENT (SKELLY AND LOY - 1975)

LEGEND

1. Should be considered
2. May be considered
3. May be considered under certain conditions
4. May be considered special situation

A. High
B. Moderate
C. Low

		Dozers	Front-End Loaders	Scrapers			Dragline	Shovel & Truck	Bucket Wheel Excavator	Front-End Loader & Truck
				Elevating	Full-Power	With Push Tractor				
Topsoil Thickness	0 -2'	1	1	1	1	1	3	-	-	1
	2'-5'	1	1	1	1	1	3	4	3	1
Haul Distance	0 -300'	1	1	2	2	1	3	-	-	-
	300'-500'	2	2	1	1	1	4	-	3	-
	500'-1000'	-	-	1	1	1	-	4	-	3
	1000'-1500'	-	-	1	1	1	-	4	-	2
	1500'-5000'	-	-	2	2	2	-	4	-	1
Flexibility Under Varied Field Conditions	good	A	A	A	A	A	A	A	B	A
	fair	A	A	A	A	A	A	B	B	B
	poor	B	B	B	B	B	A	C	C	C

1 ft. = 0.3048 m

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TABLE 12. EQUIPMENT RATING - OVERBURDEN PREPARATION (SKELLY AND LOY - 1975)

LEGEND

1. Should be considered
2. May be considered
3. May be considered under certain conditions
4. May be considered special situation

A. High
B. Moderate
C. Low

		Tractor-Ripper	Drills		
			Percussion	Rotary	Jet Piercing
Composition of Overburden	Very Consolidated	3	1	1	2
	Moderately Consolidated	1	1	1	1
	Slightly Consolidated	1	2	2	2
	Unconsolidated	1	4	4	4
Depth of Overburden	0 -30'	1	1	1	1
	30'-60'	2	1	1	1
	60'-90'	3	1	1	1
	>90'	3	1	1	2
Mobility	-	A	A	A	B
Flexibility Under Varied Field Conditions	Good	A	A	A	A
	Fair	A	A	A	A
	Poor	A	A	A	B
Hardness of Rock	Very Hard	3	1	1	2
	Moderately Hard	2	1	1	1
	Soft	1	1	1	1

1 ft. = 0.3048 meter

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TABLE 13. EQUIPMENT RATING - OVERBURDEN REMOVAL (SKELLY AND LOY - 1975)

LEGEND 1. Should be considered 2. May be considered 3. May be considered under certain conditions 4. May be considered special situation A. High B. Moderate C. Low		Dragline	Shovel	Shovel & Truck Comb.	Front-End Loaders	Dozers	Front-End Loader & Truck Comb.	Bucket Wheel Excavator	Scrapers		
									Elevating	Full Power	With Push Tractor
Thickness	0-30'	2	1	1	1	1	1	1	1	1	1
	30'-60'	1	1	1	2	2	2	1	1	1	1
	60'-100'	1	1	2	3	3	3	2	2	2	2
	>100'	1	2	3	4	4	4	2	3	3	3
Characteristics	Poor Fragmentation (Blocky-Large Breakout Force)	3	1	1	3	1	3	-	-	-	-
	Moderately Blocky	2	1	1	2	1	2	-	2	2	2
	Good Fragmentation (Low Breakout Force)	1	1	1	1	1	1	3	1	1	1
	Unconsolidated	1	1	1	1	1	1	1	1	1	1
Transport Distance	50'-150'	1	1	-	1	1	-	2	-	-	-
	150'-300'	1	-	2	1	1	-	1	-	-	-
	300'-500'	2	-	1	2	2	3	1	3	3	3
	500'-1000'	-	-	1	-	-	1	-	1	1	1
	> 1000'	-	-	1	-	-	1	-	1	1	1
Coal Seam Support Characteristics	Good (Hard)	1	1	1	1	1	1	1	1	1	1
	Moderate	1	2	2	1	1	1	2	1	1	1
	Poor (Soft)	1	4	4	2	2	2	4	1	1	1
Segregation Capability	-	A	C	A	A	B	A	A	A	A	A
Production Capability	-	A	A	B	A	A	B	A	A	A	A
Flexibility Under Varied Field Conditions	Good	A	A	A	A	A	A	B	A	A	A
	Fair	A	B	B	B	A	A	B	A	A	A
	Poor	A	C	C	C	B	B	C	B	B	B
Mobility	-	B	B	B	A	A	A	C	A	A	A

1 ft = 0.3048 meter

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TABLE 14. EQUIPMENT RATING - COAL LOADING
(SKELLY AND LOY - 1975)

LEGEND
1. Should be considered
2. May be considered
3. May be considered under certain conditions
4. May be considered special situation

A. High
B. Moderate
C. Low

		Shovels	(Rubber Tired) Front-End Loaders	High Lift (Tracks)	Backhoe	Scrapers			Bucket Wheel Excavator
						Self-Loading Full-Powered	Elevating	Under-Powered With Tractor	
Coal Seam Thickness	1'-3'	2	1	1	2	1	1	1	4
	3'-5'	1	1	1	3	1	1	1	3
	5'-10'	1	1	1	-	2	2	2	2
	10'-25'	1	2	2	-	3	3	3	1
	>25'	1	3	3	-	4	4	4	1
Fragmentation or Ease of Digging	Soft or Highly Frag.	1	1	1	2	1	1	1	1
	Moderately Frag.	1	1	1	3	2	2	2	3
	Hard or Low Frag.	1	3	2	-	4	3	4	-
Floor Conditions	Very Soft	4	1	1	2	1	1	1	3
	Moderate	1	1	1	2	1	1	1	2
	Hard	1	1	1	2	1	1	1	1
Mobility	-	B	A	A	B	A	A	A	B
Flexibility Under Varied Field Conditions	Good	A	A	A	A	A	A	A	B
	Fair	B	B	B	A	A	A	A	B
	Poor	C	B	B	B	B	B	B	C
Production Requirements	High	1	1	3	3	1	1	1	2
	Medium	1	1	1	2	1	1	1	1
	Low	1	1	1	1	1	1	1	1

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TABLE 15. EQUIPMENT RATING - REGRADING AND BACKFILLING (SKELLY AND LOY - 1975)

LEGEND

1. Should be considered
2. May be considered
3. May be considered under certain conditions
4. May be considered special situation

A. High
B. Moderate
C. Low

		Dozers	Graders	Scrapers	Draglines	Front-End Loader	Front-End Loader & Trucks
Spoil Configuration	High Peaks	1	–	2	2	3	3
	Moderate Peaks	1	3	1	2	1	2
	Low Peaks	1	1	1	2	1	1
% Spoil Rehandled	>75%	1	–	1	2	2	1
	75% - 50%	1	–	1	2	1	1
	50% - 25%	1	–	1	3	1	1
	<25%	1	2	1	–	1	1
Transport Distance	50'-150'	1	2	1	2	1	–
	150'-300'	1	–	1	2	1	–
	300'-500'	2	–	1	–	2	4
	500'-1000'	–	–	1	–	–	1
	>1000'	–	–	1	–	–	1
Final Surface Contour	Flat & Smooth	1	1	1	–	3	–
	Flat & Rough	1	2	1	2	2	–
	Steep & Smooth	1	–	–	–	3	–
	Steep & Rough	1	–	–	2	2	–

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TABLE 16. EQUIPMENT RATING - REVEGETATION
(SKELLY AND LOY - 1975)

LEGEND 1. Should be considered 2. May be considered 3. May be considered under certain conditions 4. May be considered special situation A. High B. Moderate C. Low		Airplane	Hydroseeder	Broadcast	Drill Seeding	Disc & Row Planting	Hand Planting or Spreading	Mulching Machine
Acreage	<3	-	1	1	1	1	3	1
	3-10	-	1	1	1	1	4	1
	10-25	-	1	2	1	1	-	1
	25-50	3	1	2	2	2	-	1
	>50	2	1	3	4	3	-	1
Seed Mixture	Grasses	1	1	1	1	3	3	-
	Crops	-	3	2	1	1	3	-
	Seedlings	-	-	-	-	1	2	-
Mulching	-	-	1	-	-	-	3	1
Fertilizer & Lime	-	1	1	3	1	1	3	-

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TABLE 17. EQUIPMENT RATING - COAL HAULING (SKELLY AND LOY - 1975)

LEGEND 1.Should be considered 2.May be considered 3.May be considered under certain conditions 4.May be considered special situation A.High B.Moderate C.Low		Trucks					Scrapers					
		Off Road			Highway							
		End Dump	Bottom Dump	Trailer End Dump	End Dump	Trailer End Dump	Full Power	Elevating	Under Powered w/Tractor	Conveyor	Train	Pipeline
Material	Rough-Blocky	1	-	1	1	1	-	-	-	-	1	-
	35 inch size	1	-	1	1	1	-	-	-	2	1	-
	24 inch size	1	2	1	1	1	1	1	1	1	1	-
	Fines	1	1	1	1	1	1	1	1	1	1	4
Length of Haul	300'-500'	1	2	2	2	2	2	2	2	4	-	4
	500'-1000'	1	1	1	1	1	1	1	1	4	-	4
	1000'-1500'	1	1	1	1	1	1	1	1	4	-	4
	1500'-5000'	1	1	1	1	1	1	1	1	3	-	4
	5000'-10,000'	1	1	1	1	1	3	3	3	2	2	3
	10,000'-15,000'	1	1	1	1	1	-	-	-	1	1	3
	>15,000'	1	1	1	1	1	-	-	-	1	1	2
Condition of Road	Good	1	1	1	1	1	1	1	1	-	-	-
	Wet, Soft	1	1	1	2	2	3	3	3	-	-	-
Type of Haulage Roads	Mine Roads Only	1	1	1	2	2	1	1	1	-	-	-
	Combination: Mine & Public	2	2	2	1	1	4	4	4	-	-	-
Daily Production Rate	High	1	1	1	1	1	1	1	1	1	1	4
	Medium	1	1	1	1	1	1	1	1	2	2	4
	Low	1	1	1	1	1	1	1	1	-	-	-
Flexibility Under Varied Conditions	Good	A	A	A	A	A	A	A	A	B	C	-
	Fair	A	A	A	A	A	A	A	A	B	C	-
	Poor	A	A	A	B	B	B	B	B	C	C	-
Maximum Adverse Grade	+3%	1	1	1	1	1	1	1	1	1	1	1
	+5%	1	2	2	1	2	1	2	2	1	2	1
	+10%	2	3	3	2	3	2	2	2	1	-	2
	+15%	3	-	-	3	-	2	3	3	1	-	2
	+20%	-	-	-	-	-	3	4	4	2	-	3
	>+20%	-	-	-	-	-	4	-	-	3	-	3

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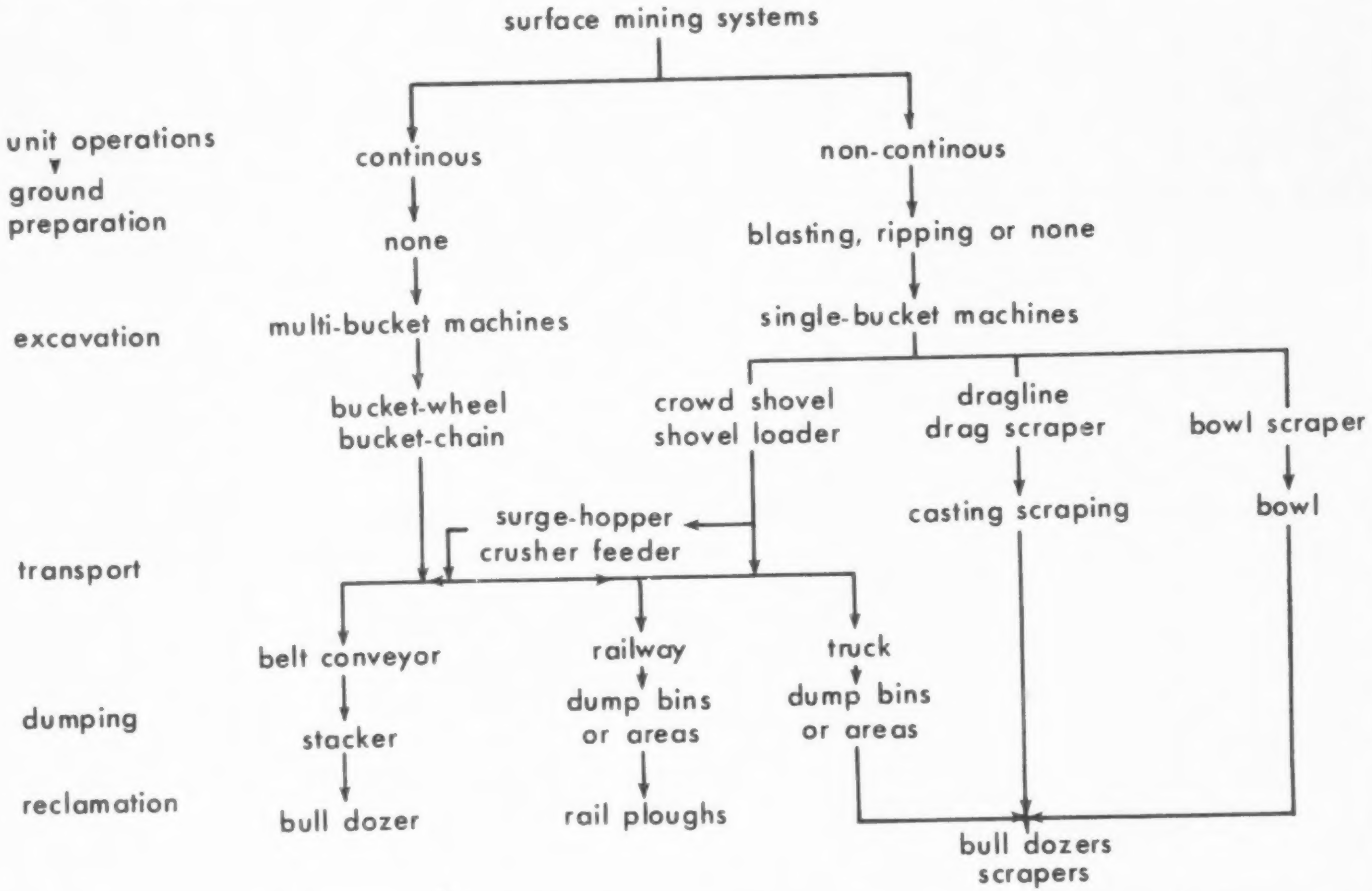
surface mining systems
unit operations
ground preparation
excavation
transport
dumping
reclamation
continous
none
multi-bucket machines
bucket-wheel
bucket-chain
non-continous
blasting, ripping or none
single-bucket machines
crowd shovel
shovel loader
dragline
drag scraper
bowl scraper
casting scraping
bowl
surge-hopper
crusher feeder
belt conveyor
stacker
bull dozer
railway
dump bins
or areas
rail ploughs
truck
dump bins
or areas
bull dozers
scrapers

Figure 25. Equipment application for surface mining systems (Atkinson, 1971).

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TABLE 18. SINGLE-PHASE STRIPPING POSSIBILITIES

Number	Approach Name	General Method	Equipment Used Excavation	Equipment Used Transport
(1)	Dragline Overcasting	Overcasting	Dragline	Dragline
(2)	Shovel Overcasting		Stripping Shovel	Stripping Shovel
(3)	Bucketwheel Stripping	Passover	Bucket-wheel Excavator	Belt Conveyor (Stacker)
(4)	Loader-Conveyor Stripping*		Front-end Loader	Belt Conveyor
(5)	Shovel-Conveyor Stripping*		Loading Shovel	Belt Conveyor
(6)	Bulldozer Stripping	Crossover	Bulldozer	Bulldozer
(7)	Scraper Stripping		Scraper	Scraper
(8)	Loader Stripping		Front-end Loader	
(9)	Dragline-truck Stripping	Haul-Around	Dragline	Truck
(10)	Shovel-truck Stripping		Loading Shovel	Truck
(11)	Loader-truck Stripping		Front-end Loader	Truck
(12)	Dredging*		Dredge	Dredge Pipeline

* These methods are not commonly practiced in coal in the Eastern United States.

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TABLE 19. MULTIPLE-SEAM STRIPPING POSSIBILITIES* (RAMANI, 1968)

Upper Seam Method	Feasible Lower Seam Method
1	1, 2, 6, 7, 8, 9, 10, 11
2	1, 2, 6, 7, 8, 9, 10, 11
3	1, 2, 6, 7, 8, 9, 10, 11
4	1, 2, 6, 7, 8, 9, 10, 11
5	1, 2, 6, 7, 8, 9, 10, 11
6	6, 7, 8, 9, 10, 11
7	6, 7, 8, 9, 10, 11
8	6, 7, 8, 9, 10, 11
9	6, 7, 8, 9, 10, 11
10	6, 7, 8, 9, 10, 11
11	6, 7, 8, 9, 10, 11
12	1, 2, 6, 7, 8, 9, 10, 11

* Refer to Table 18 for the description of the numbered method.

Dragline Overcasting

Figure 26 is a photograph of a 34 cu m (45 cu yd) dragline in western Pennsylvania. Up to 26 m (85 ft) of overburden are cast across an 18 m (60 ft) pit to exploit an 0.5 m (1.67 ft) coal seam. Figure 27 is a line diagram of the largest dragline operation. A 168 cu m (220 cu yd) machine removes 36.5 to 40 m (120 to 130 ft) of overburden to exploit a 1.2 m (4 ft) coal seam in a pit 76 m (250 ft) wide. Small dragline operations are common in the hilly terrain of Pennsylvania. Figure 28 shows a large dragline pit with other surface mining equipment.

Shovel Overcasting

Shovel overcasting is similar to dragline overcasting but incorporates a stripping shovel instead of a dragline. Figure 29 shows the second largest stripping shovel working in 45 m (150 ft) of overburden in Ohio. Figure 30 shows details of a shovel pit.

Bucket-Wheel Stripping

Stripping with a bucket-wheel excavator (BWE) alone is not practiced in the Eastern United States. The BWE can effectively handle soft material. Its performance is poor in hard material or strata that has to be fragmented. The applications common in the study area of this report incorporate a two-phase stripping system, in which the upper, unconsolidated material is removed by the BWE, and the lower, hard strata which must be fragmented is handled by either a stripping shovel or a dragline.

BWE-Dragline Stripping

Figure 31 is a line diagram, and Figure 32 is a photograph of an operation of this type in Illinois. Both machines work at the same bench elevation, 0 to 20 m (0 to 65 ft) below the surface. Up to 37.0 m (120 ft) of overburden are removed, 20.0 m (65 ft) by the BWE and 17.0 m (55 ft) by the dragline. The pit width is 28.0 m (90 ft), and the coal seam is 0.91 m (3 ft) thick.

BWE-Shovel Stripping

Figure 33 is a line diagram and Figure 34 is a photograph of an operation of this type in southern Illinois. The BWE cuts its own bench from 0 to 16.8 m (0 to 54 ft) below the surface. The shovel operates in the pit at a depth of 14.0 m (46 ft) below the BWE bench. Pit width is 33.5 m (110 ft) and the coal seam is 1.37 m (4 to 5 ft) thick.

Smaller Scale Approaches

The preceeding discussions have centered on the largest scale mines, typical of Illinois, Indiana, and Ohio. In the east, area mines are usually of such limited dimensions that large-scale and fixed equipment are not used. A variety of approaches are taken in smaller-scale operations, approaches which incorporate virtually every combination of equipment and methods.

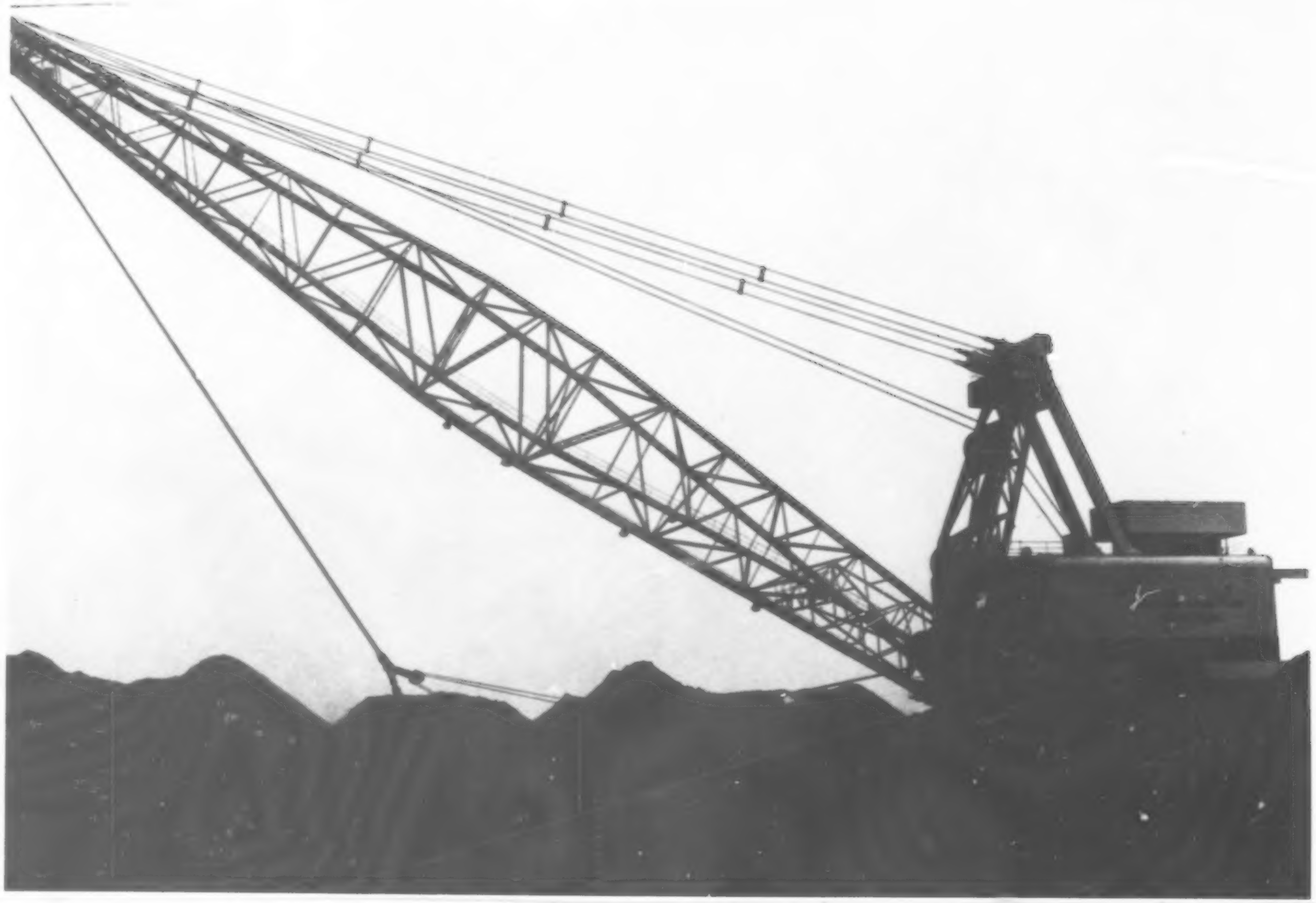
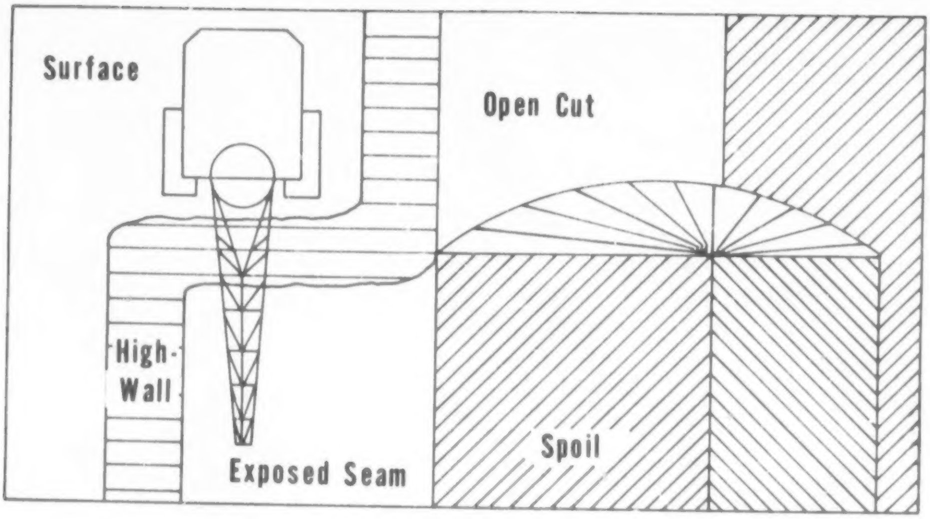
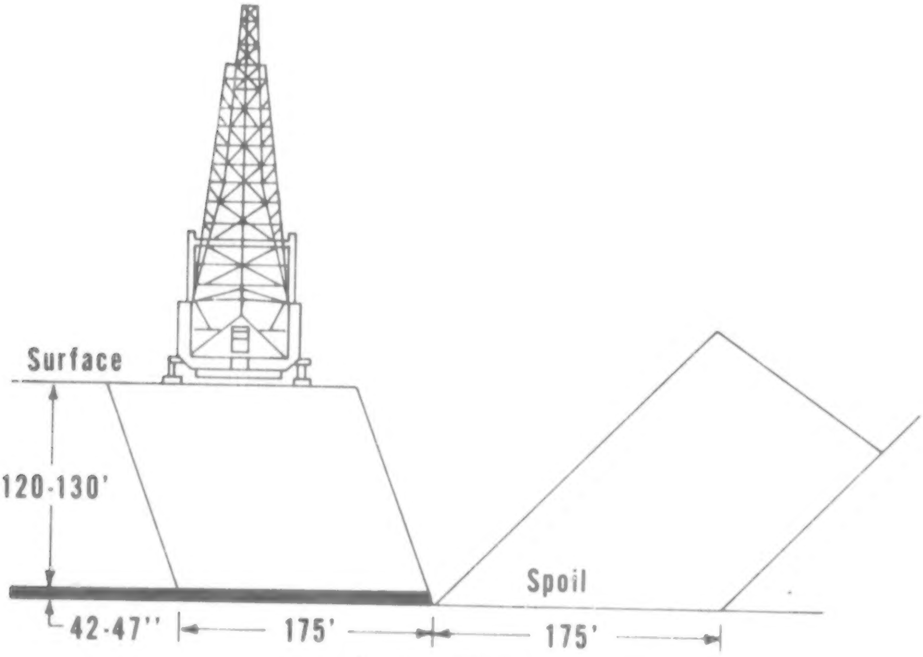


Figure 26. A 45-cubic-yard dragline at work in a Pennsylvania coal operation.


Surface
Open Cut
High-Wall
Exposed Seam
Spoil

Plan View


Surface
120-130'
42-47''
175'
175'
Spoil

Section View

Figure 27. Pit layout at Bucyrus-Erie 4250-W operation (Stefanko, Ramani, and Ferko, 1973).

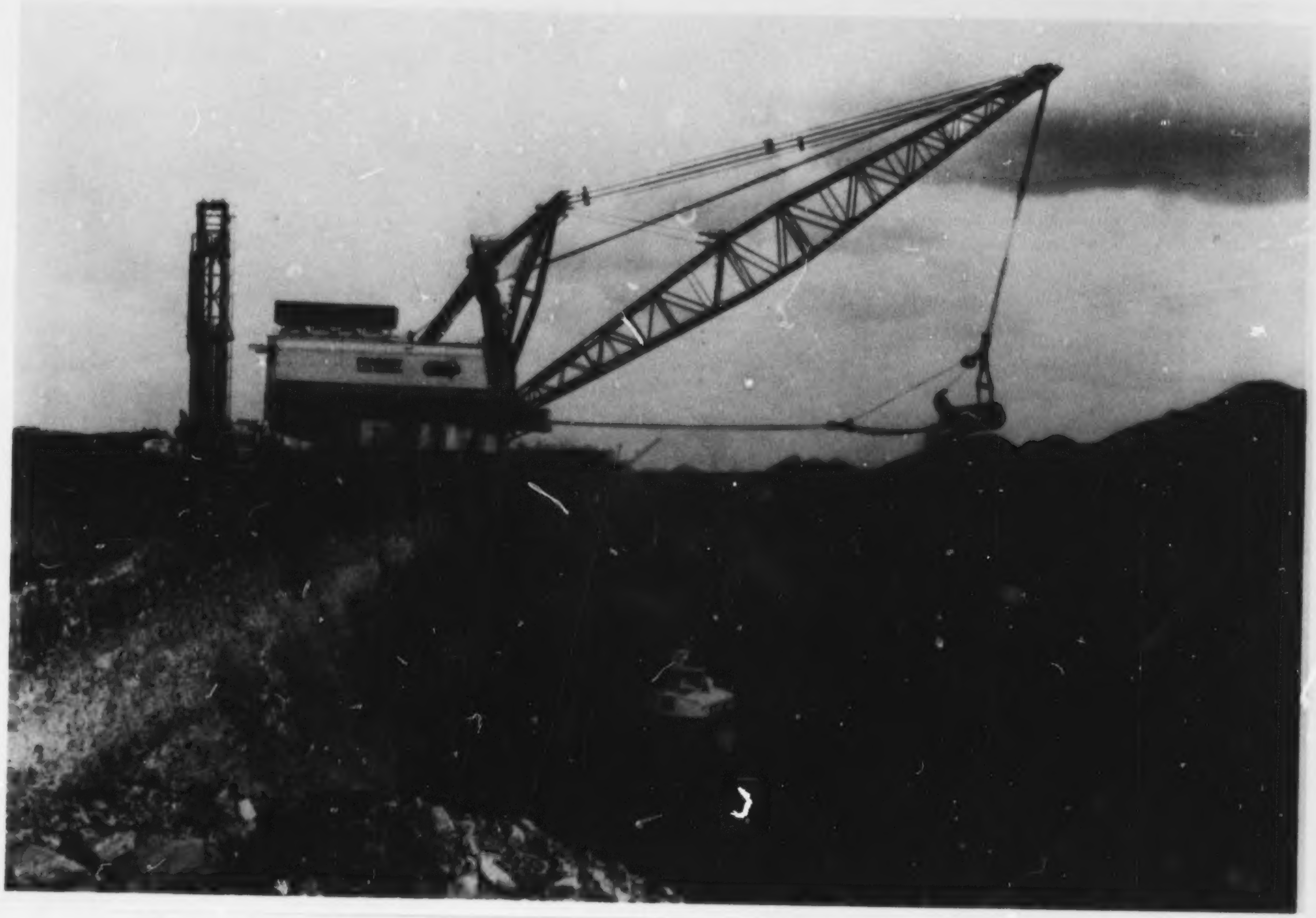


Figure 28. A 90-cubic-meter dragline with a 90-meter boom. Note the coal-loading shovel and truck in the pit (Ramani and Grim, 1978).

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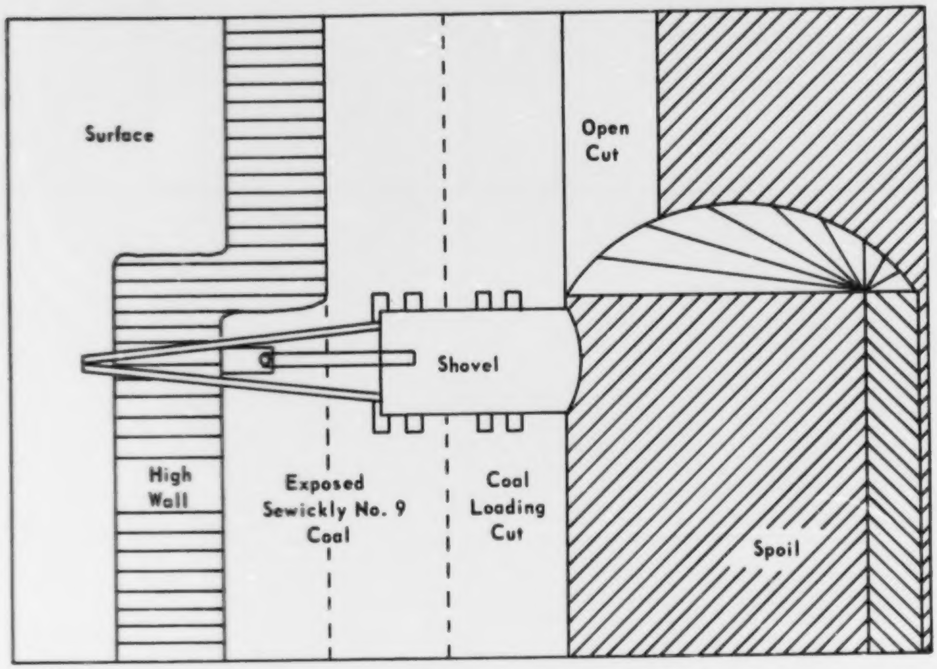
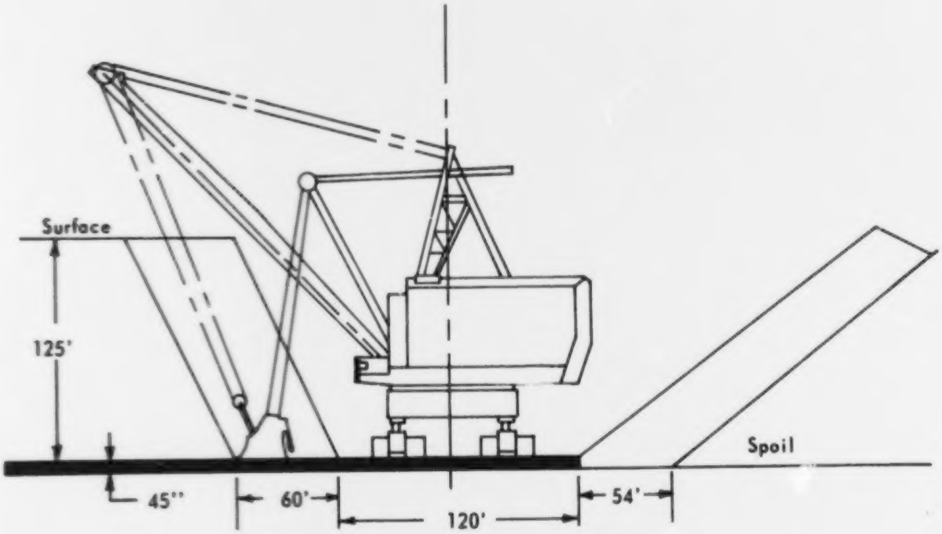

Surface
Open Cut
Shovel
High Wall
Exposed Sewickly No. 9 Coal
Coal Loading Cut
Spoil

Plan View


Surface
125'
Spoil
45''
60'
120'
54'

Section View

Figure 29. Plan and section views of a Bucyrus-Erie 1950-B pit (Stefanko, Ramani and Ferko, 1973).

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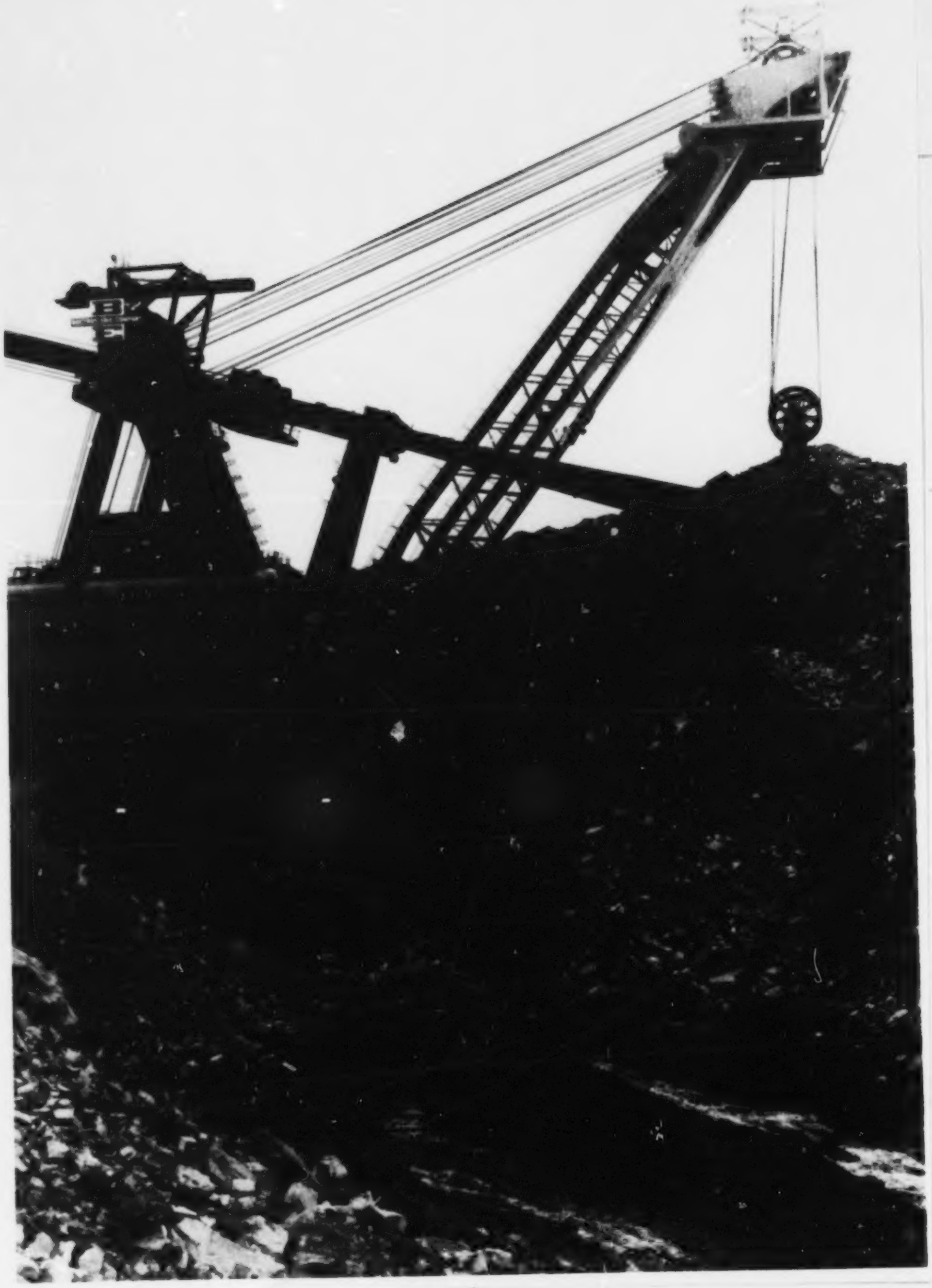

Figure 30. Details of a shovel pit.

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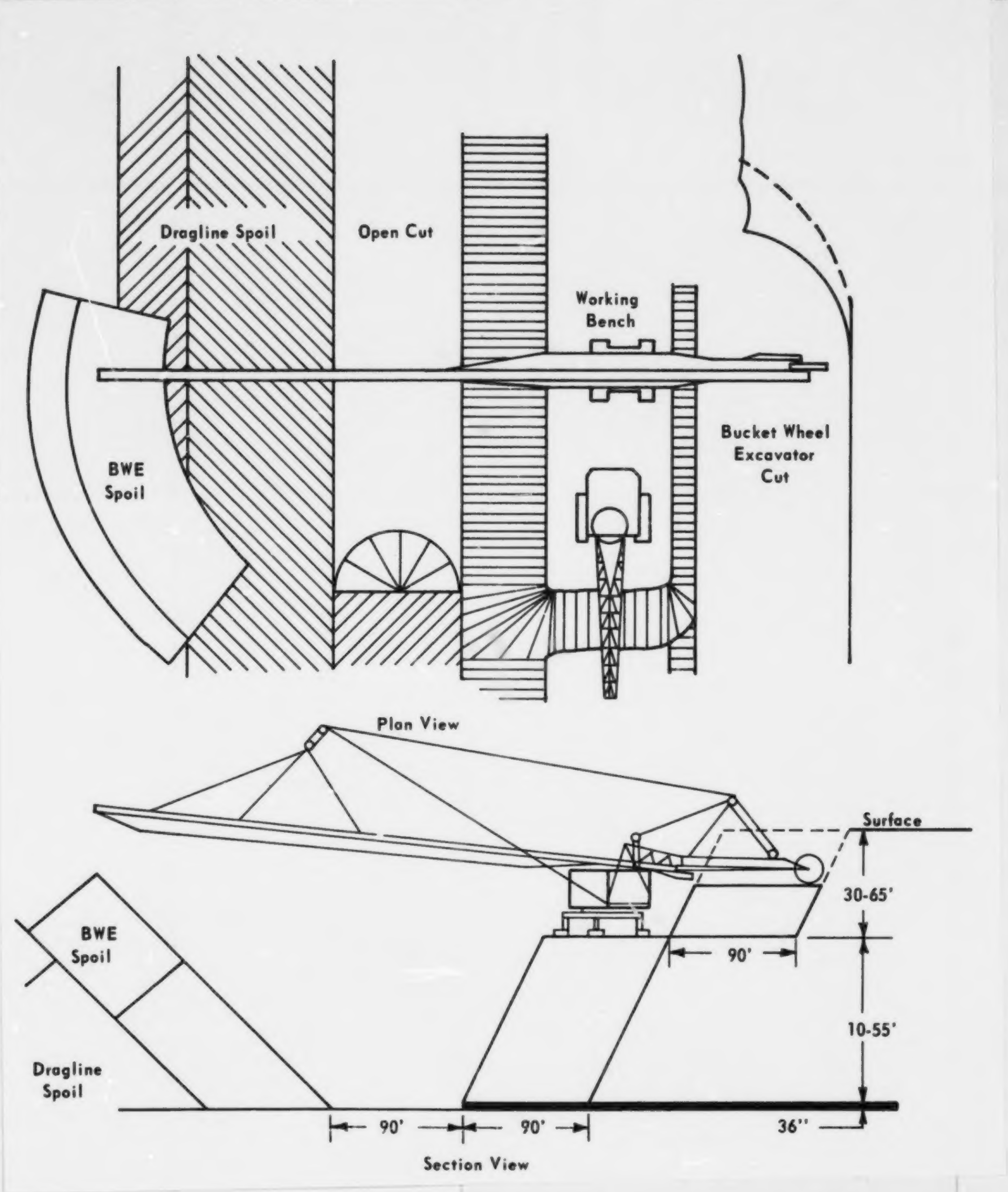
Dragline Spoil
Open Cut
Working Bench
BWE Spoil
Bucket Wheel Excavator Cut
Plan View
Surface
30-65'
90'
10-55'
BWE Spoil
Dragline Spoil
90'
90'
36"
Section View

Figure 31. Plan and section views of a dragline and bucket-wheel excavator in tandem operation (Stefanko, Ramaṇi, and Ferko, 1973).

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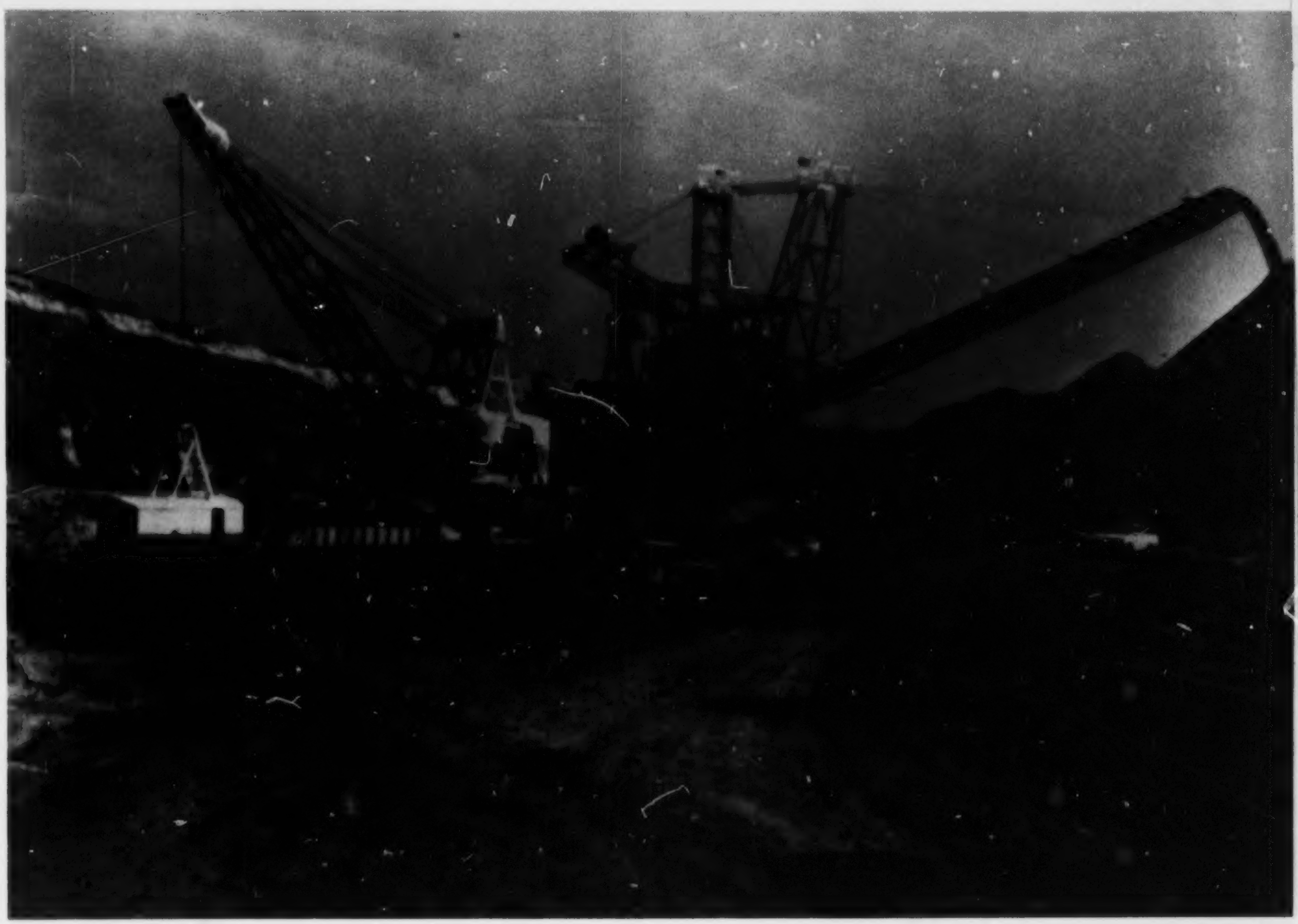

Figure 32. Bucket-wheel excavator, dragline, coal-loading shovel, trucks, and a dozer ripper are employed in this operation (Ramani and Grim, 1978).

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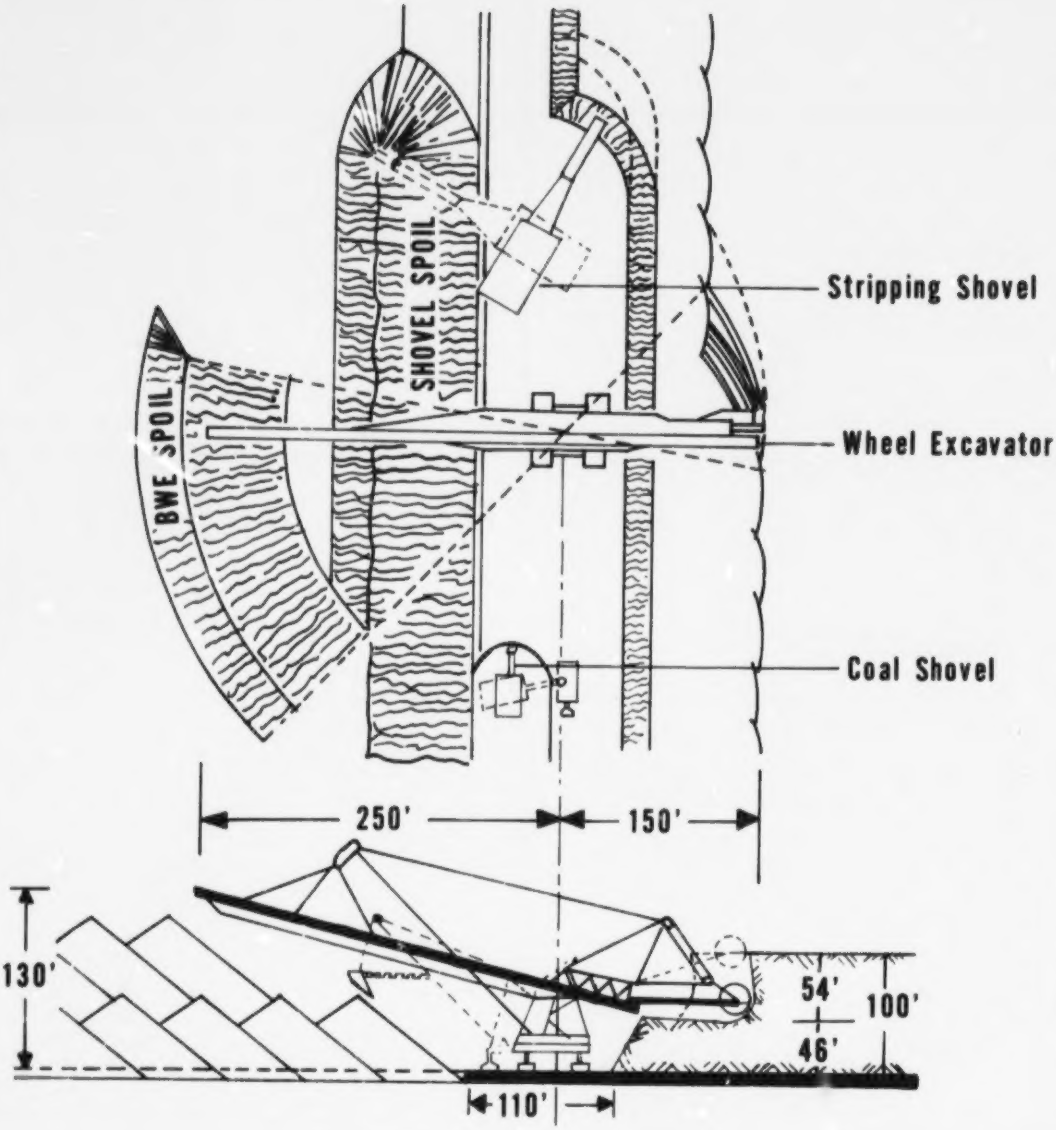
SHOVEL SPOIL
BWE SPOIL
Stripping Shovel
Wheel Excavator
Coal Shovel
250'
150'
130'
54'
100'
46'
110'

Figure 33. Pit layout at a shovel and bucket-wheel excavator tandem operation in Illinois (Ramani, 1968).

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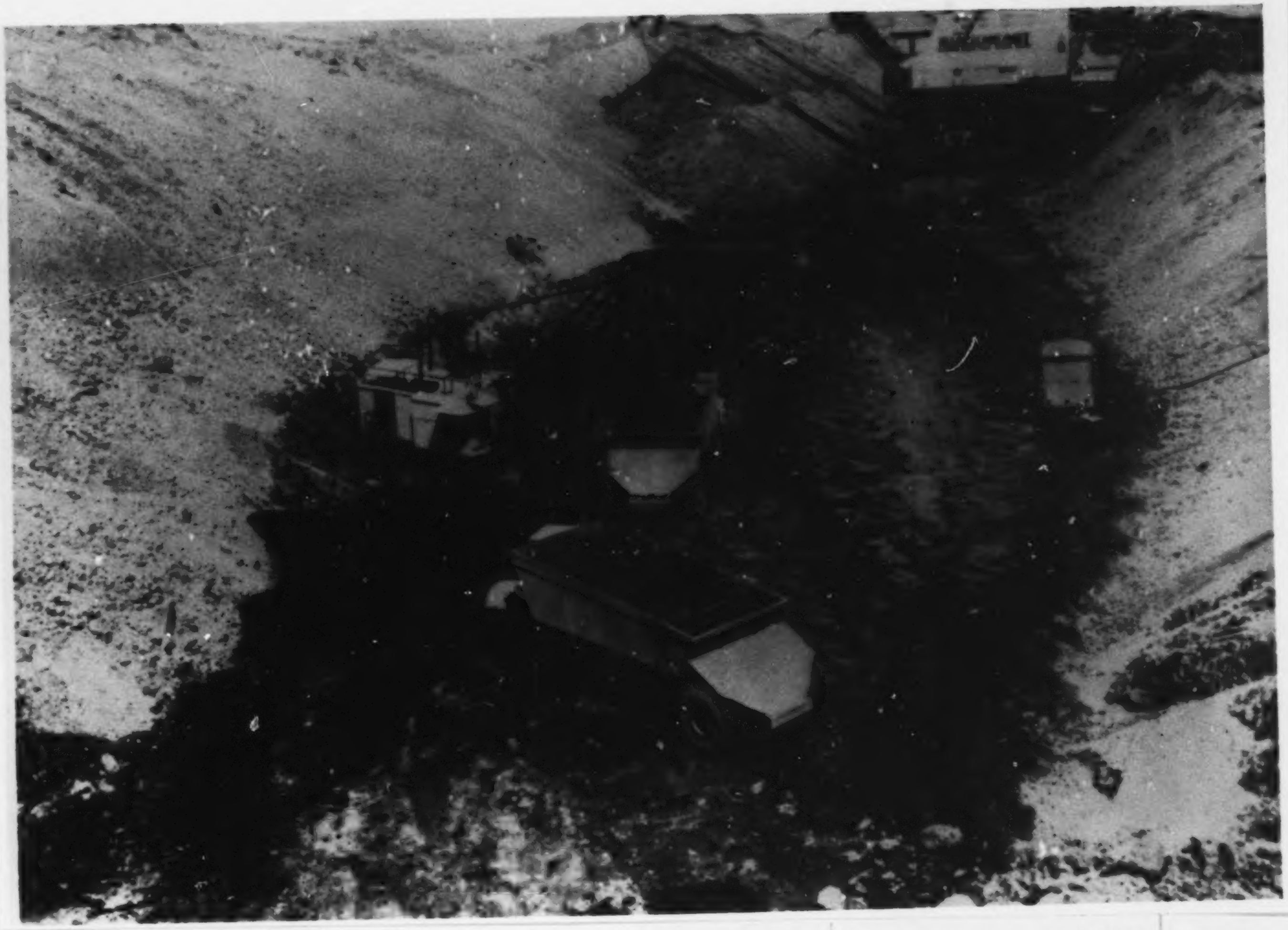

Figure 34. An operation in Illinois using bucket-wheel excavator, stripping shovel, loading shovel and trucks (Ramani and Grim, 1976).

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CONTOUR MINING

Contour mining is practiced where an outcrop of coal occurs on a hillside. The mining may progress into the hillside only to a certain point, after which there is too much overburden on top of the coal for economic removal. The mining may proceed around the hillside, however, at the given elevation. Successive cuts are removed along the contour line, hence the name contour mining. There are several methods of contour mining such as outslope disposal, outslope reduction, box cut, block cut, haulback or step blockcut, modified area, mountaintop removal, and head-of-hollow fill methods. For all practical purposes outslope disposal of spoil is no longer prevalent or allowed. The outslope disposal method is sometimes referred to as "shoot 'n shove" mining. As shown in Figure 19, the approach is to fragment the overburden by blasting from the outcrop into the stripping limit, and simply to shove the material over the edge. Outslope disposal contour mining (Figure 19) was the traditional approach but is legally limited today, due to its degrading effect on the environment of the mining area. Soil stability, erosion, sedimentation, water pollution, etc., are some of the problems that are enhanced by this method.

Outslope reduction methods are based on the theory that the stability of the outslope spoil bench could be enhanced if the slope of that bench were reduced and the spoil spread over a larger area. Grim and Hill (1974) discuss the various outslope reduction methods.

Outslope reduction was accepted in the recent past as an approach to mining in mountainous terrain and was often practiced as a means of reclaiming abandoned contour strip mines. It can still be used to reduce the slope of existing spoil pile or refuse bank areas. However, federal legislation and some state regulations have made clear the undesirability of any spoil disposal downslope from the outcrop or beyond the solid edge of the bench on slopes over 20 degrees. Therefore, these methods will not be discussed here. The interested reader is referred to Grim and Hill (1974) for detailed discussion of many of these methods. On the other hand, extended presentation of methods that have been developed to decrease or eliminate the outslope disposal of overburden are presented.

The Head-of-Hollow Fill Technique

Head-of-hollow and valley fills basically consist of storing the overburden material in narrow, steep-sided valleys according to a well engineered plan. Due to the swelling of material when broken, careful analysis of the cut-and-fill problem associated with the method is required. This method represents a controllable alternative to outslope disposal. Although the disposal is technically downslope and outside the solid bench, a specific, controlled situation is developed for such disposal.

While a wide range of valleys or hollows is acceptable for head-of-hollow fill purposes, v-shaped, narrow, steep-sided valleys located near the tops of ridges and free from connections to subsurface drainage (e.g., old underground mine openings, seeps, and springs) are preferred. The size of the valley selected for filling is a function of the amount of spoil which

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will be generated in the mining operation. The objective is to develop a valley fill which completely fills the prepared valley, all the way to the head.

While there are many references regarding the construction of a valley fill, they are either general in nature or specific in certain aspects relating explicitly to requirements of the state of operation. Therefore, dimensions and specifications for valley fill construction that are provided here should be interpreted only as generally indicative of current practices. Several important procedures are followed in adapting the head-of-hollow fill and mountaintop removal methods (to be discussed later in this section) to a particular area. Figure 35 presents a drawing of the head-of-hollow fill method. The head-of-hollow selected for filling is scalped of vegetative cover. Topsoil is removed and stored for later use. Check dams or silt control structures are built downstream from the hollow fill (Figure 36). The natural drainway in the hollow is deepened. The minimum width recommended for the drain is 4.56 meters. A French drain is built in this drainway with rocks no less than 0.36 meters in size in any one dimension (Figures 37 and 38). The fill is built in layers, 1.2 meters thick, beginning at the toe of the valley (Figure 39). The overburden material is deposited in uniform horizontal layers and then compacted with haulage equipment. Each layer of fill is built so that the face of the fill has an outer slope no steeper than two horizontal to one vertical, and usually has crowned terraces (Figure 40). Some operators have graded the face to approximately 22 degrees from the horizontal, eliminating crowned terraces. However, long slopes without interruptions, such as diversion ditches, are more prone to excessive erosion from surface runoff; therefore, where the face of the fill is not stepped, diversion ditches at a minimum of every 15 meters vertical height of the fill is recommended. Revegetation of the hollow-fill face, usually with hydroseeding, is done as soon as the fill height increases.

Block Cut

In the block-cut method, the general progress of mining is along a contour. However, mining is accomplished by taking successive blocks along the contour which may be from one to several cuts deep. Figure 41 shows how these successive block cuts are carried along the contour. The advantages of this method are that it allows overburden spoiling in the pit and on the bench and that each cut is worked to the ultimate limit of the stripping ratio once it is opened. In this second consideration, the small operator often finds the previously mentioned, against-the-contour advance of each block cut easier to handle on an economic basis.

Several extensions of block-cut mining are practiced. They have a number of different names, such as modified block cut, haulback, pit storage, and put-and-take. Figure 42 is a line drawing of the sequencing of cuts, and Figures 43 and 44 explain the stripping and backfilling phases.

Basic block-cut mining is frequently confused with these extensions. In this method, the width of each block is the same. The depth each block is carried into the hillside may vary, however, as each block is carried to

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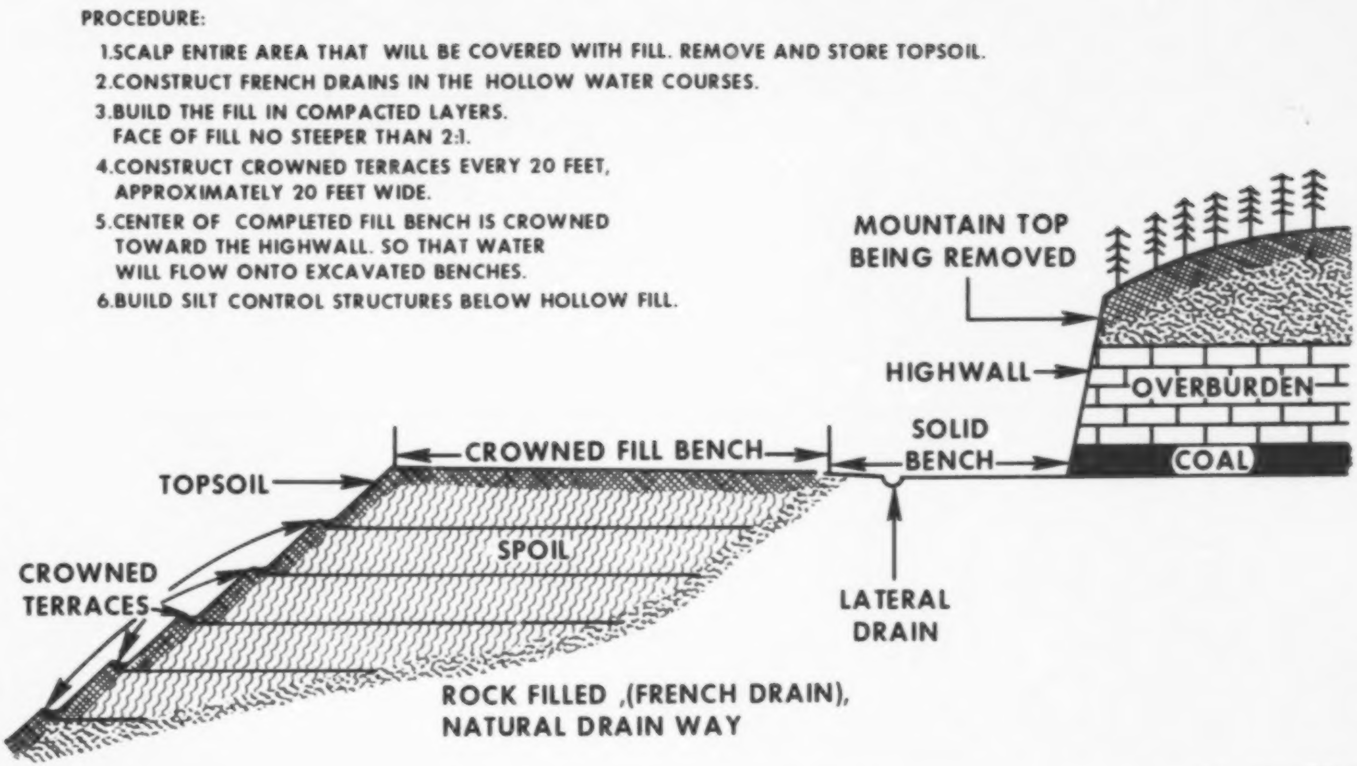

PROCEDURE:

1. SCALP ENTIRE AREA THAT WILL BE COVERED WITH FILL. REMOVE AND STORE TOPSOIL.
2. CONSTRUCT FRENCH DRAINS IN THE HOLLOW WATER COURSES.
3. BUILD THE FILL IN COMPACTED LAYERS. FACE OF FILL NO STEEPER THAN 2:1.
4. CONSTRUCT CROWNED TERRACES EVERY 20 FEET, APPROXIMATELY 20 FEET WIDE.
5. CENTER OF COMPLETED FILL BENCH IS CROWNED TOWARD THE HIGHWALL. SO THAT WATER WILL FLOW ONTO EXCAVATED BENCHES.
6. BUILD SILT CONTROL STRUCTURES BELOW HOLLOW FILL.

MOUNTAIN TOP BEING REMOVED
HIGHWALL
OVERBURDEN
SOLID BENCH
COAL
CROWNED FILL BENCH
TOPSOIL
SPOIL
CROWNED TERRACES
LATERAL DRAIN
ROCK FILLED ,(FRENCH DRAIN), NATURAL DRAIN WAY

Figure 35. Head-of-hollow fill (Grim and Hill, 1974).

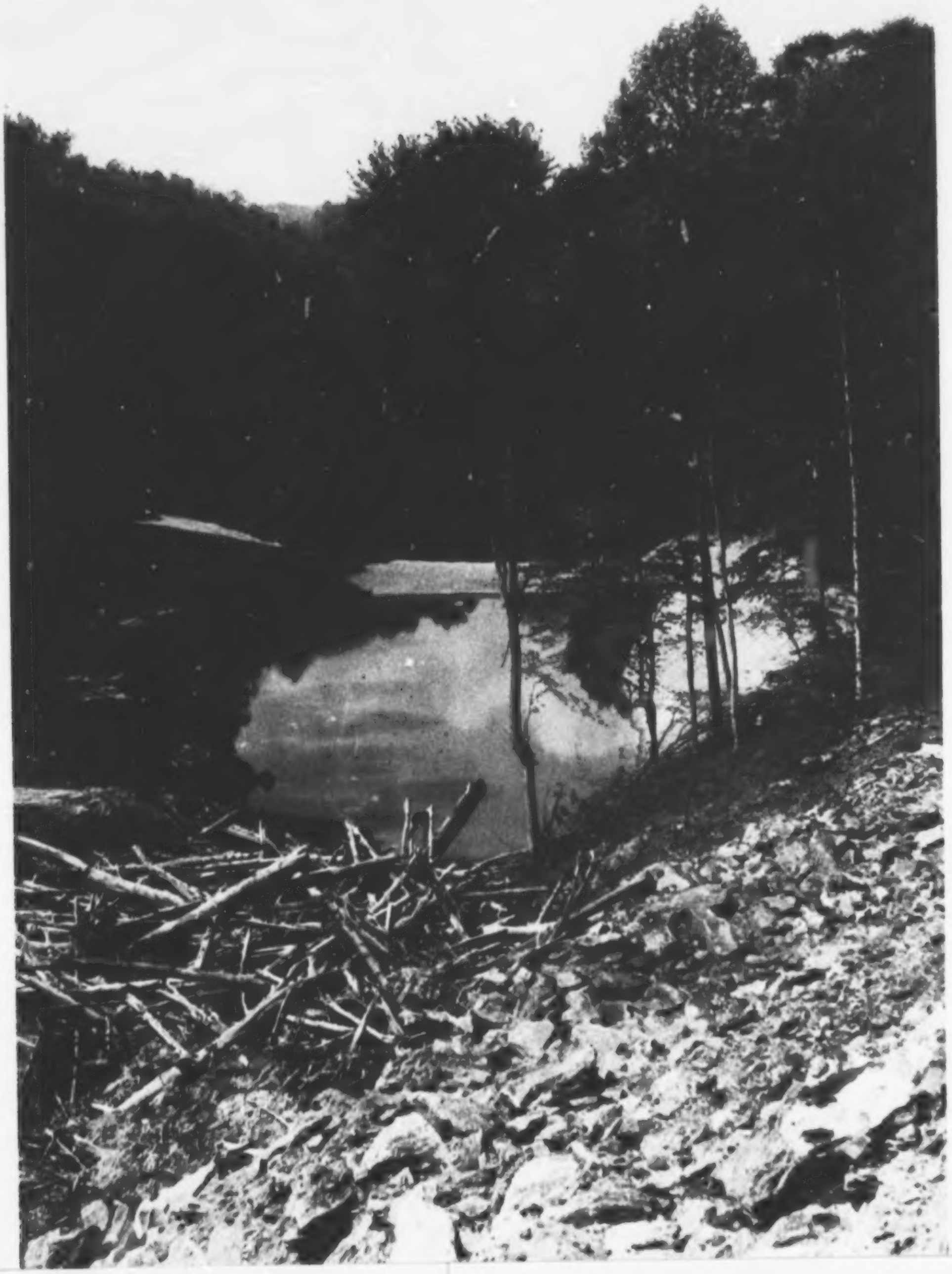

Figure 36. A check dam, downstream from a hollow fill.

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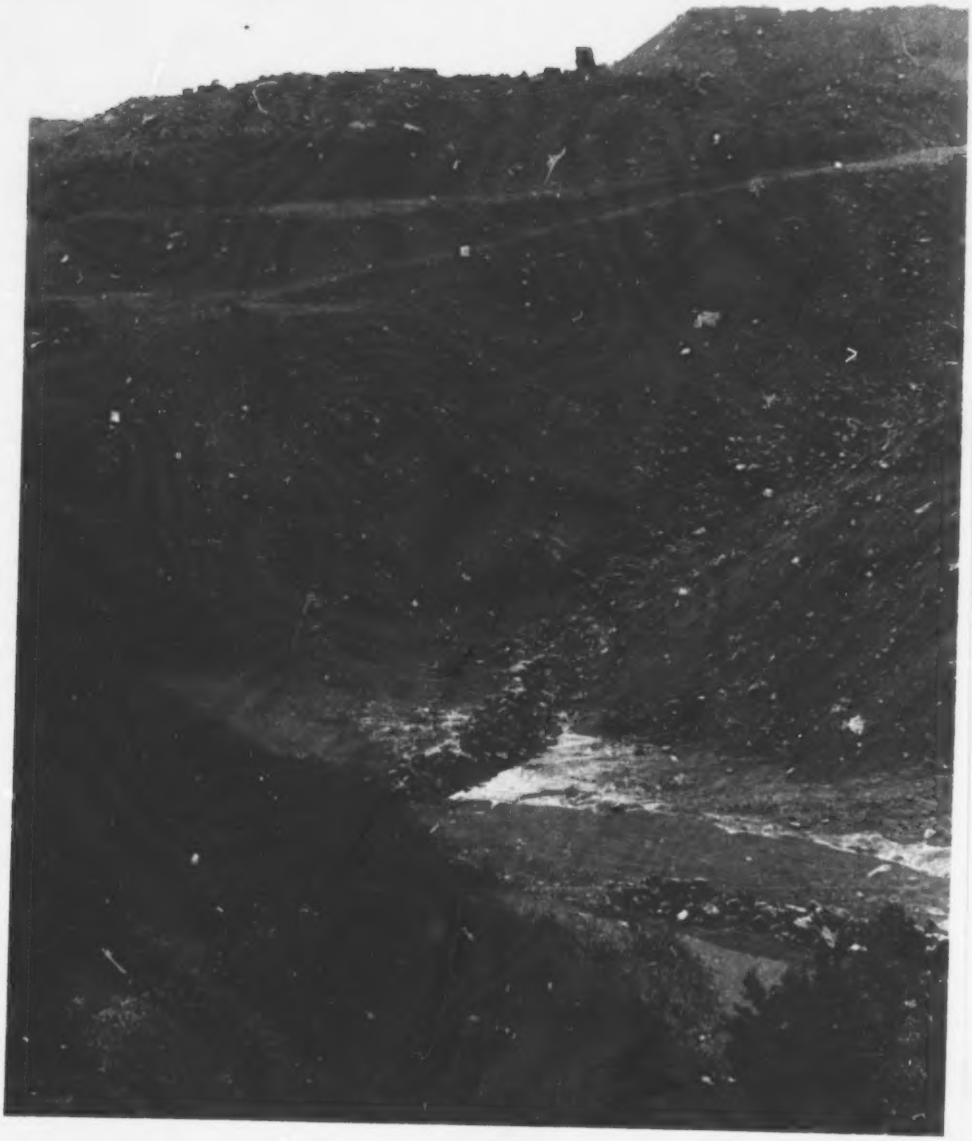

Figure 37. A distant view of the French drain.

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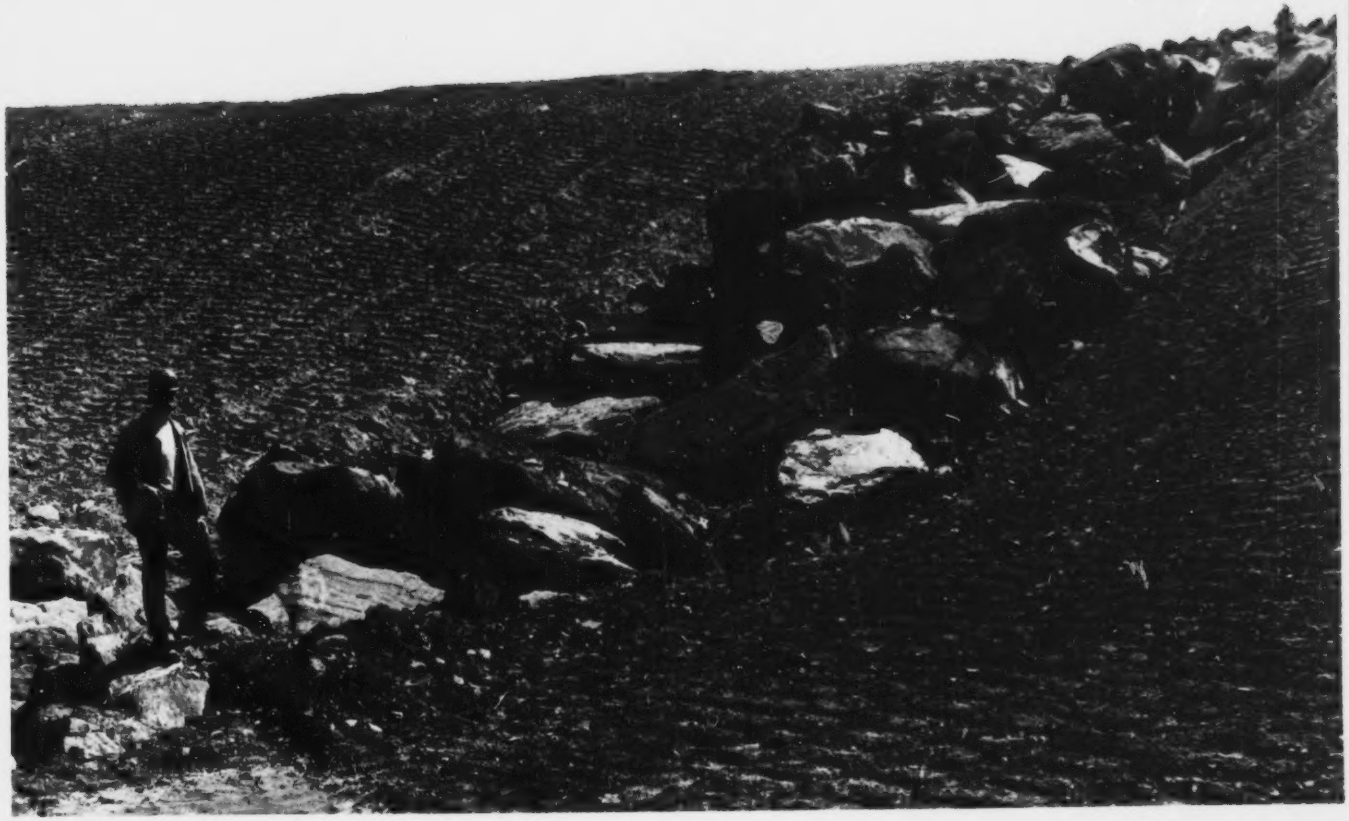

Figure 38. The size and gradation of the rock that form the core of the French drain is an important design consideration.

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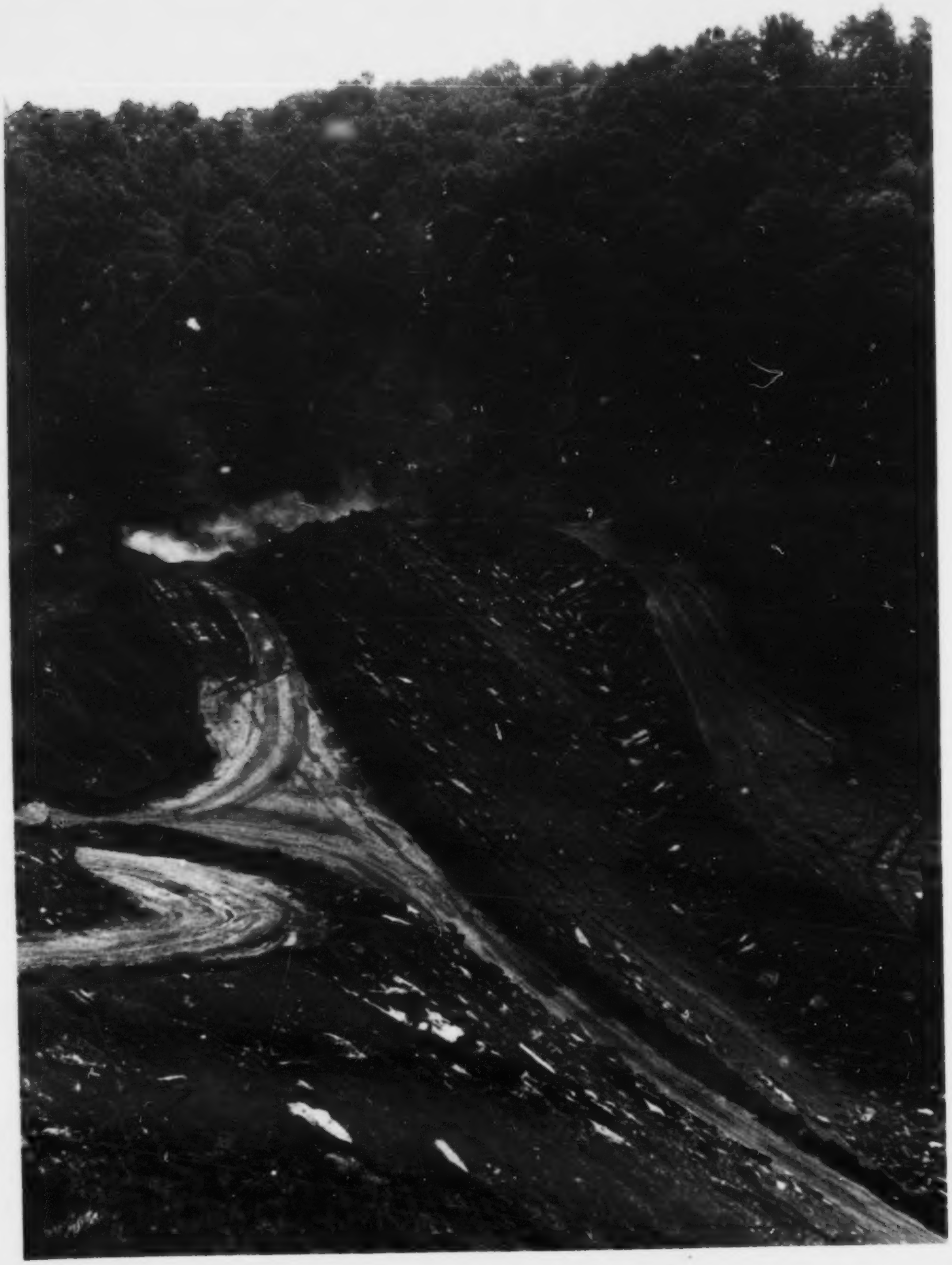

Figure 39. The valley fill is built from the bottom up in layers.

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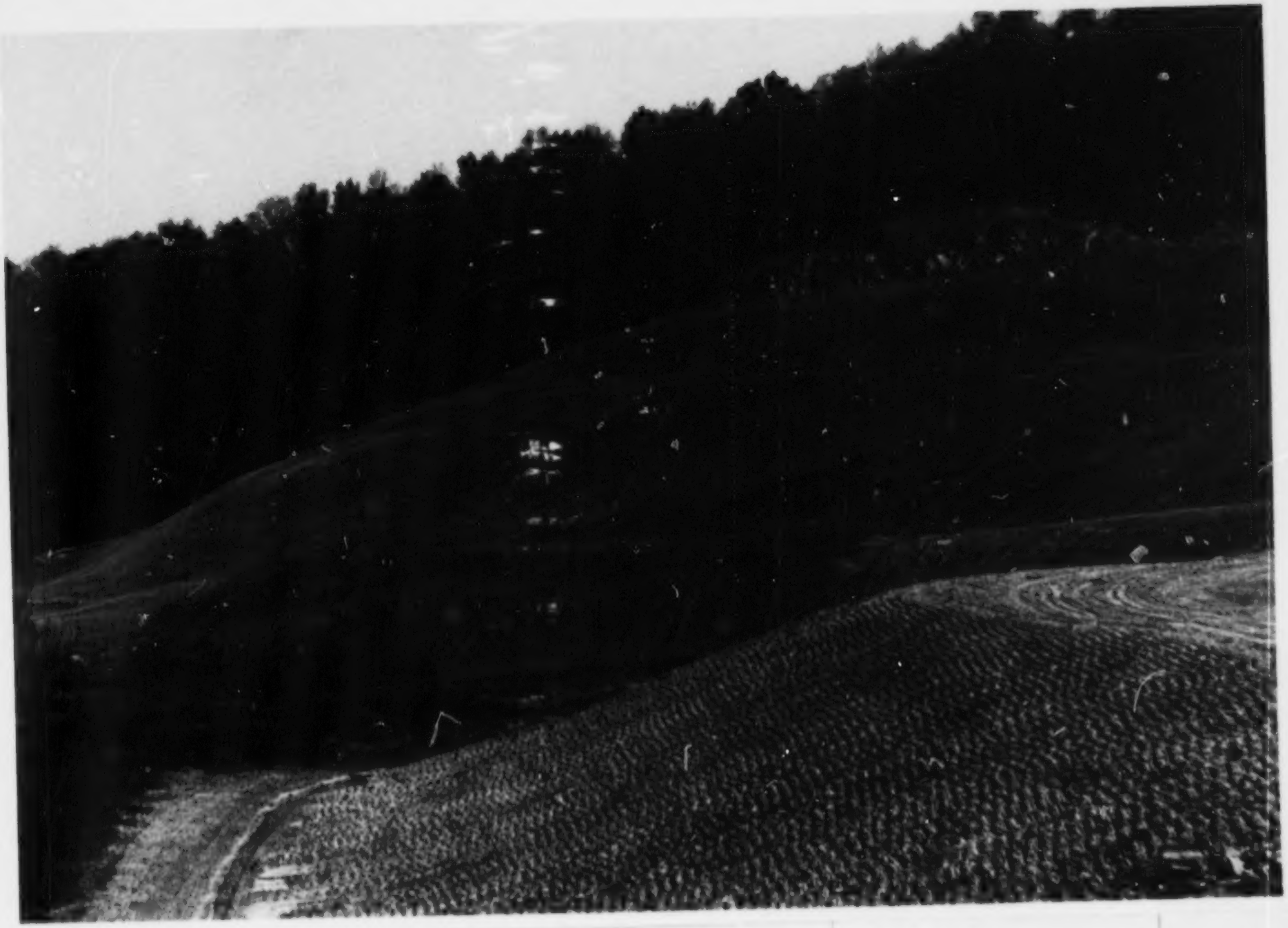

Figure 40. A completed head-of-hollow fill.

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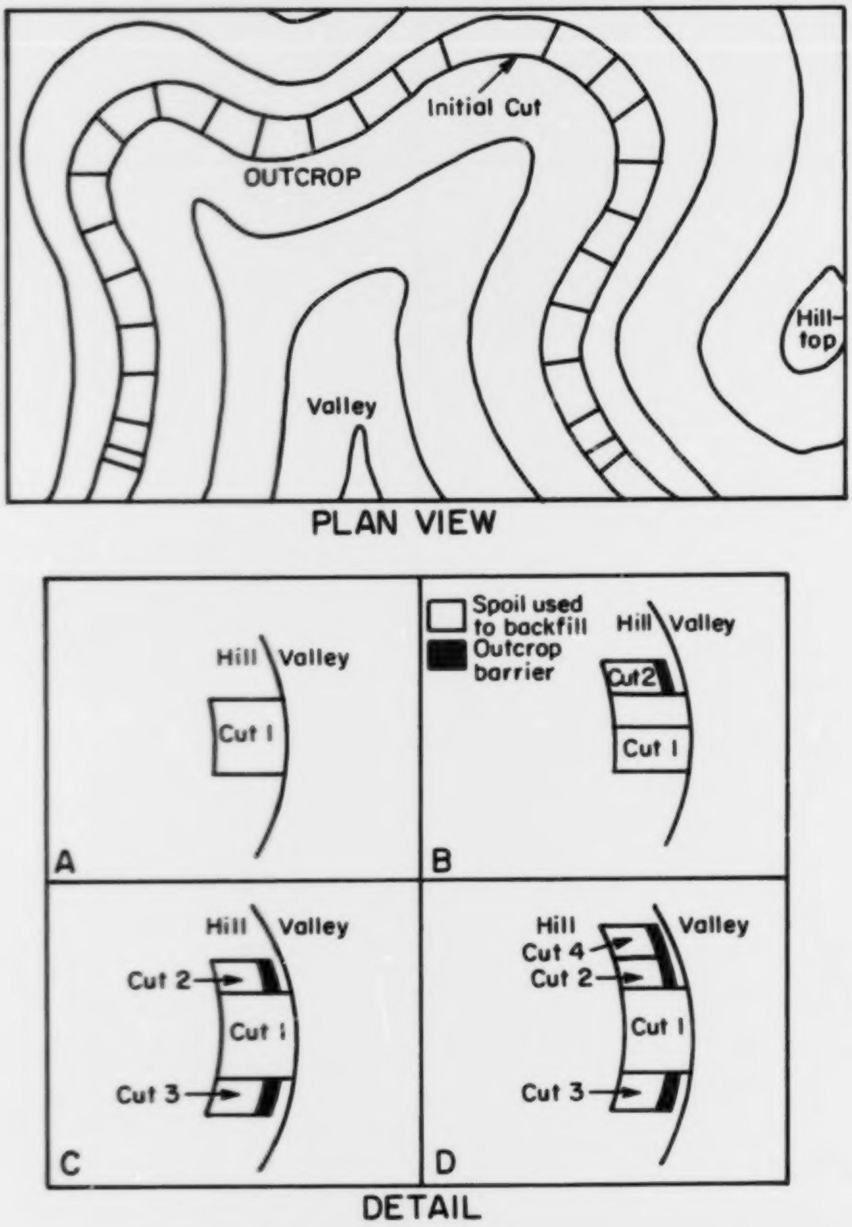
Initial Cut
OUTCROP
Hill-top
Valley
PLAN VIEW
Hill
Valley
Cut 1
A
Spoil used to backfill
Outcrop barrier
Hill
Valley
Cut 2
Cut 1
B
Hill
Valley
Cut 2
Cut 1
Cut 3
C
Hill
Valley
Cut 4
Cut 2
Cut 1
Cut 3
D
DETAIL

Figure 41. Contour mining around a hillside with the block system (Saperstein and Secor, 1973; Maneval, 1972).

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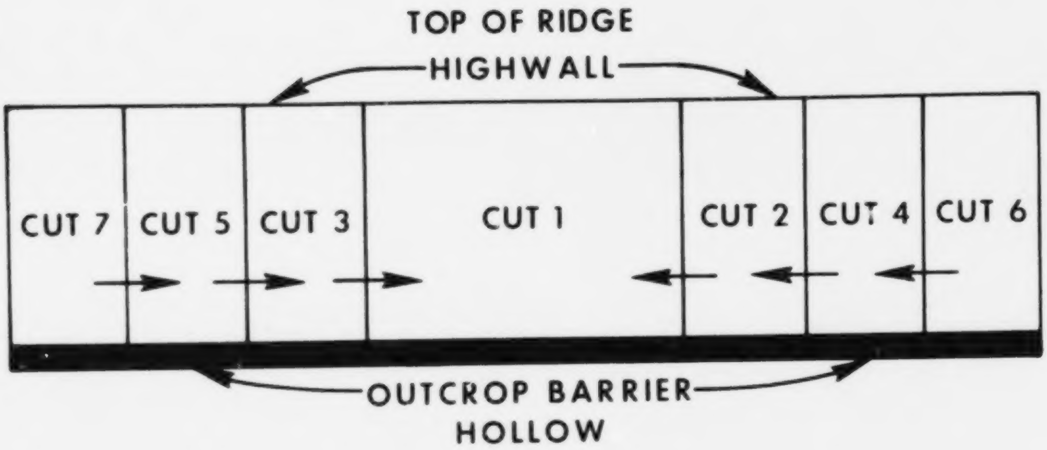
TOP OF RIDGE
HIGHWALL
CUT 7
CUT 5
CUT 3
CUT 1
CUT 2
CUT 4
CUT 6
OUTCROP BARRIER
HOLLOW

PROCEDURE:

1. SCALP FROM TOP OF HIGHWALL TO OUTCROP BARRIER, REMOVE AND STORE TOPSOIL.
2. REMOVE AND DISPOSE OF OVERBURDEN FROM CUT 1.
3. PICK UP COAL, LEAVING AT LEAST A 15 FOOT UNDISTURBED OUTCROP BARRIER.
4. MAKE SUCCESIVE CUTS AS NUMBERED.
5. OVERBURDEN IS MOVED IN THE DIRECTION, AS SHOWN BY ARROWS, AND PLACED IN THE ADJACENT PIT.
6. COMPLETE BACKFILL AND GRADING TO THE APPROXIMATE ORIGINAL CONTOUR.

Figure 42. Block-cut method (Grim and Hill, 1974).

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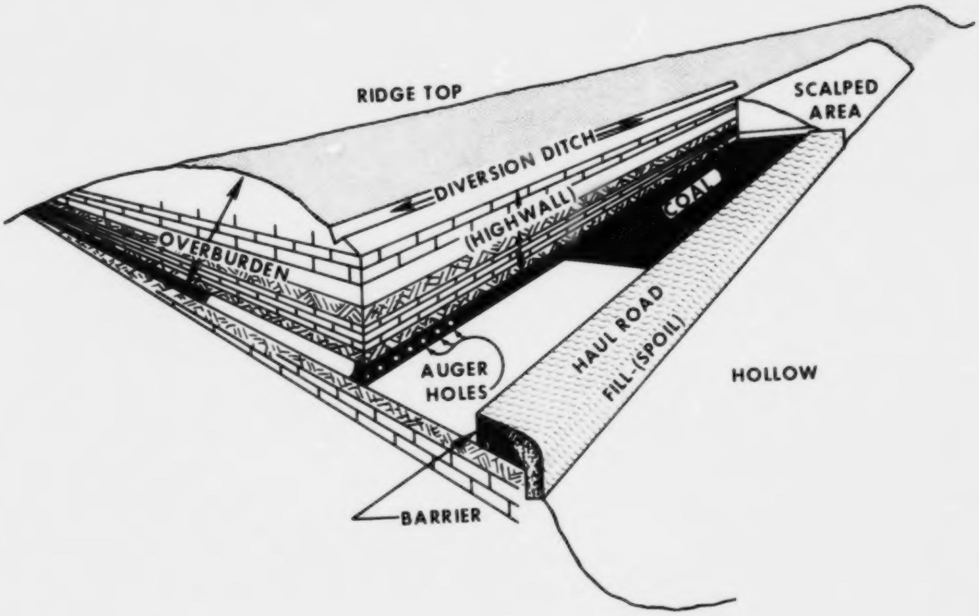
RIDGE TOP
SCALPED AREA
DIVERSION DITCH
(HIGHWALL)
COAL
OVERBURDEN
AUGER HOLES
HAUL ROAD
FILL-(SPOIL)
HOLLOW
BARRIER

Figure 43. Block-cut method: stripping phase (Grim and Hill, 1974).

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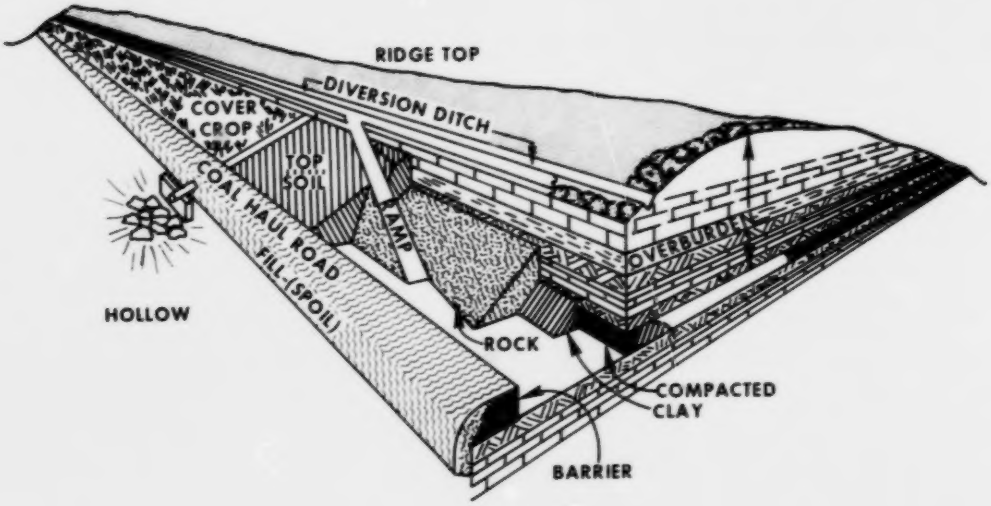
RIDGE TOP
DIVERSION DITCH
COVER CROP
TOP SOIL
AMP
OVERBURDEN
COAL HAUL ROAD
FILL (SPOIL)
HOLLOW
ROCK
COMPACTED CLAY
BARRIER

Figure 44. Block-cut method: backfilling phase (Grim and Hill, 1974).

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its maximum allowable stripping ratio. One advantage is that the dragline can work, for example, cuts 3 and 4 from one location. Another is that the dragline can work uniformly along the highwall with short move time. Aside from the initial cut, the spoil is placed in a previous excavation, which limits rehandling and is environmentally preferable.

The modified block cut and haulback extensions capitalize on the pit-spoiling aspects of block cut. In these, the first cut is sized larger than the others to assure that the volumes of broken and swollen overburden from the remaining cuts will be exactly equal to the mined-out volume. Therefore, once the spoil of the first cut is disposed of in a valley fill or other suitable manner, the remaining spoil can be stored in the pit. The difference in modified block cut and haulback is that modified block cut proceeds in one direction along the contour and may utilize any of several techniques for overburden handling, while haulback proceeds in both directions from the initial cut and utilizes load and carry or truck transport of overburden. The advantage of the haulback refinement of pivoting about a central location seems to be proven through actual practice in both methods. Haulback is unique, as it allows continuous overburden removal, coal removal, augering and reclamation in one pit.

Block-cut methods have been accepted by industry and government alike. Saperstein and Secor (1973, 1977) found that the block-cut methods can be cost-competitive with any other contour mining system.

One key to success in the block-cut methods is the determination of the ultimate stripping ratio. Given this ultimate ratio, the block depths must then be established. For modified block cut to haulback, extensive premining planning is necessary as additional considerations are brought in to assure that the proper allowances for swell have been developed and the first cut and valley fill have been properly sized. A rigorous analysis of the overburden handling unit operation is required. Many combinations of excavation and transport equipment are possible. The sizing and logistical configuration of many alternatives must be considered if an optimum is to be found. Grim and Hill (1974) list the following advantages and disadvantages of the block-cut method:

Advantages

1. Since the method is not limited by the slope of the area, currently available equipment can be used. Further, the method can be used in multiple-seam applications where unit operations must be closely concentrated.
2. Less coverage is disturbed than in other mining methods. The mined-out areas are completely backfilled, eliminating downslope spoil disposal and the final highwall.
3. Reclamation costs are lower, and revegetation costs are reduced. Also, overburden segregation is enhanced.
4. The problems associated with water quality control (acid mine drainage, siltation, and sedimentation), are reduced, since toxic or material which is difficult to handle can be selectively placed

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in isolation as the block cuts are filled. Fewer sediment control structures are required and the size of the disturbed-area drainage system is smaller.

Disadvantages

1. Extensive premining planning and scheduling is required. The location and size of the initial box cut must be precisely determined. Further, investment costs for overburden transport equipment are high, discouraging application in small operations.
2. The drilling and blasting processes are complicated and time-consuming even though they are standard approaches. Overburden must be controlled and fragmentation must be matched to the equipment used in excavation and transport.
3. The actual long-term environmental consequences are not known or documented.
4. Additional recovery techniques must be carried out concurrent with initial mining. Grim and Hill specifically mention augering. Any future recovery of coal from the seam by a surface approach requires complete removal of the spoil that has been replaced.

The most attractive aspect of the block-cut technique is that the recovery of the mineable resource, and the restoration and rehabilitation of the surface are an integral part of the total mining process.

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Mountaintop Mining

Mountaintop mining is an adaption of area mining methods to contour-mining situations. The two approaches used are modified area mining and mountaintop-removal mining. Where coal lies under ridges, mountaintops, and knobs, there is usually a stripping-ratio limit to the depth into the hillside to which mining may proceed. Mountaintop methods are, however, the exception in which the overall stripping ratio is taken as the guide rather than local and ultimate maximum stripping ratios. It is based on the analysis that area mining methods, even at a higher average stripping ratio, allow the recovery of more coal (by removal of all the coal in a given tract) at a lower total cost per ton, due to economies of scale in operation, than is possible in contour mining. Figures 45 through 48 explain the sequence of operations.

These methods are applied where a coal seam is present on opposite sides of a hill or ridge to a great extent, and the overburden is sufficiently thin (less than 50 m [165 ft]) across the coal, to allow area mining. The method is applicable to previously contour-mined ridges and hills as well as to coal seams in virgin areas. When restoration is to approximate original contour, the best methods to follow are the dragline or shovel-casting approaches of area mining, and the technique is modified area. When the mountaintop is restored as essentially flat land, the best approaches incorporate load-and-carry, bulldozer, or loader-truck techniques, and the method is designated as mountaintop removal.

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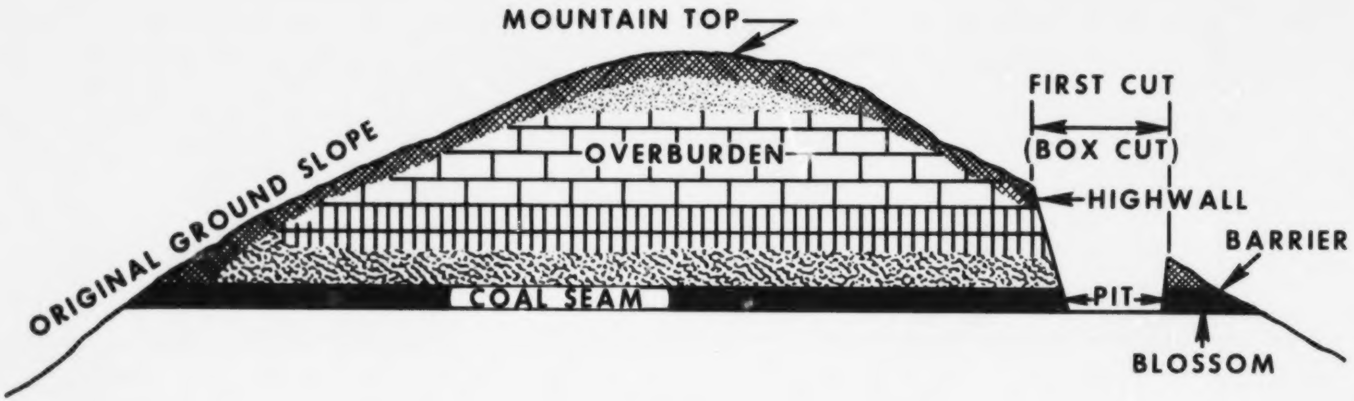
MOUNTAIN TOP
FIRST CUT
(BOX CUT)
OVERBURDEN
HIGHWALL
BARRIER
ORIGINAL GROUND SLOPE
COAL SEAM
PIT
BLOSSOM

Figure 45. Mountaintop removal method: first cut (box cut) (Grim and Hill, 1974).

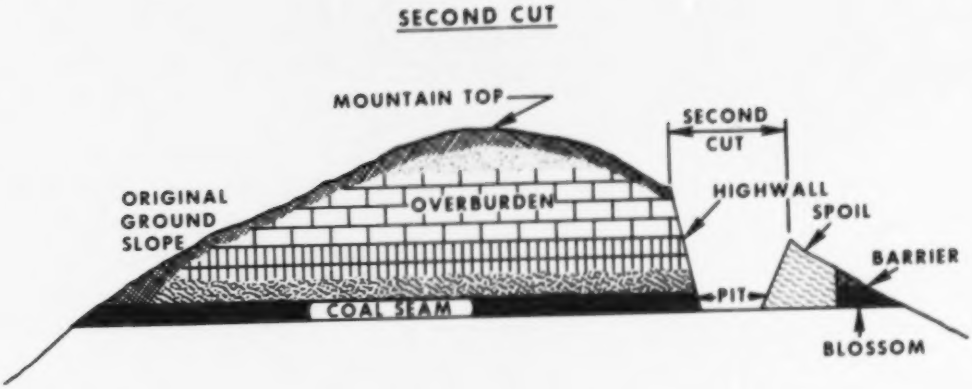
SECOND CUT
MOUNTAIN TOP
SECOND CUT
ORIGINAL GROUND SLOPE
OVERBURDEN
HIGHWALL
SPOIL
BARRIER
PIT
COAL SEAM
BLOSSOM

Figure 46. Mountaintop removal method: second cut (Grim and Hill, 1974).

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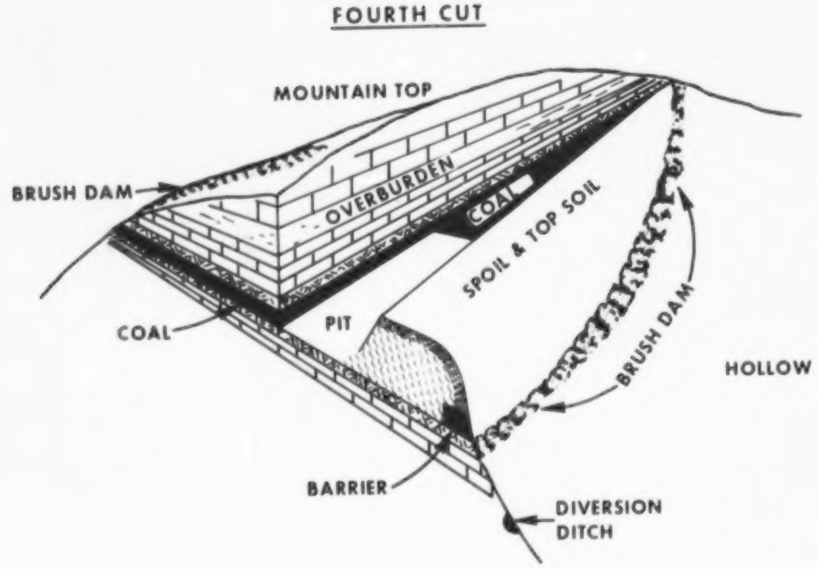
FOURTH CUT
MOUNTAIN TOP
BRUSH DAM
OVERBURDEN
COAL
SPOIL & TOP SOIL
COAL
PIT
BRUSH DAM
HOLLOW
BARRIER
DIVERSION DITCH

Figure 47. Mountaintop removal method: fourth cut (Grim and Hill, 1974).

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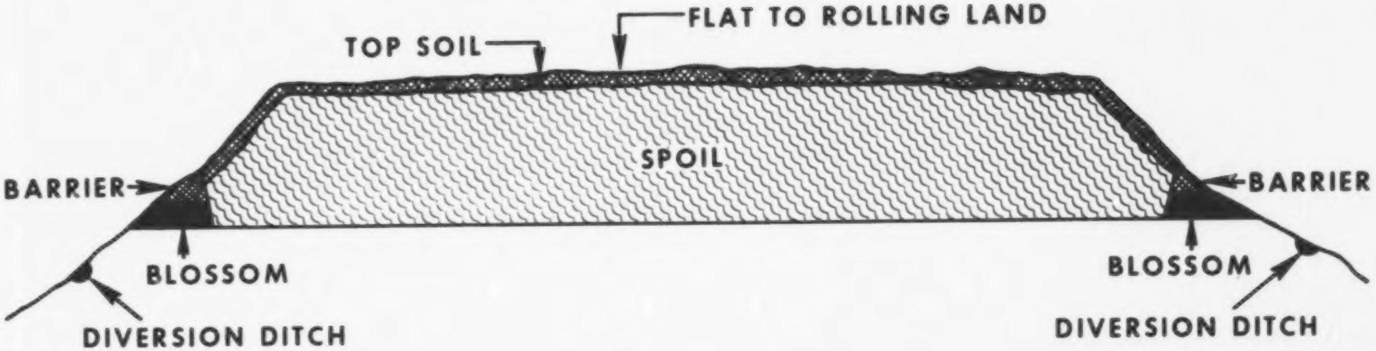
FLAT TO ROLLING LAND
TOP SOIL
SPOIL
BARRIER
BARRIER
BLOSSOM
BLOSSOM
DIVERSION DITCH
DIVERSION DITCH

Figure 48. Mountaintop removal method: mountaintop after final grading and topsoiling (Grim and Hill, 1974).

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Modified area mining is practiced in areas such as western Pennsylvania and western Kentucky. The topography and the requirements of restoration to the original contour are two reasons for its practice. The spoil in modified areas is placed in the mine excavation. The value of the land is taken as better in original contour than in revised contour.

Mountaintop removal is practiced most extensively in West Virginia. Mountaintop removal incorporates the use of valley fills to which a significant portion of the spoil is committed. For example, by this method, it is possible to create from 50 hectares of coal land 75 to 100 hectares of "flat" restored land surrounded by stable and controlled outslopes. In most cases, such gently rolling land is worth much more in its state of revised contour than in its previous sloping state. Near Welch, West Virginia, 30.4 hectares (75 acres) of mountaintop removal land were donated as a high school building site, and the remaining land can be used for subdivision. In this area, residential land is worth as much as $98,684 per hectare ($40,000 per acre).

The steps in the mountaintop removal method are essentially the same as in valley fill and are as follows:

6-1/2"

1. Determine the postmining volume of overburden. Given this volume, determine the amount that will be spoiled in the mined-out area and the amount that will be spoiled in valley fills.
2. Prepare the required valley fill areas (Figure 49).
3. Scalp (clear and grub) areas for temporary storage of topsoil.
4. Scalp the hilltop. Remove topsoil to temporary storage areas.
5. Initiate mining with a typical area-mining trench cut along one edge of the property. Usually a 5 m (16.5 ft) barrier of blossom coal is left along the outcrop edge.
6. Continue to mine relatively parallel to the first cut across the mountaintop. In the modified area, the dragline or shovel spoils most of the overburden in the previous pit to the approximate original contour. In mountaintop removal, much of the spoil in initial cuts is placed in valley fills by the mobile equipment fleets; when these valleys are completely filled, spoiling is carried out on the mountaintop.
7. Grade the mountaintop and fills to approximate original contour (modified area) or to essentially flat, revised contour (mountaintop removal).
8 Restore topsoil, amend and revegetate (Figure 50).

Mountaintop mining methods offer the same general advantages as the block cut and other practices that do not include outslope-disposal. The determination of the size and location of valley fills and the sequencing of cutting and filling operations are crucial for success. The increased value of the flat land created due to this method, may enable surface mining of coal which heretofore were recoverable by either underground or auger-mining techniques.

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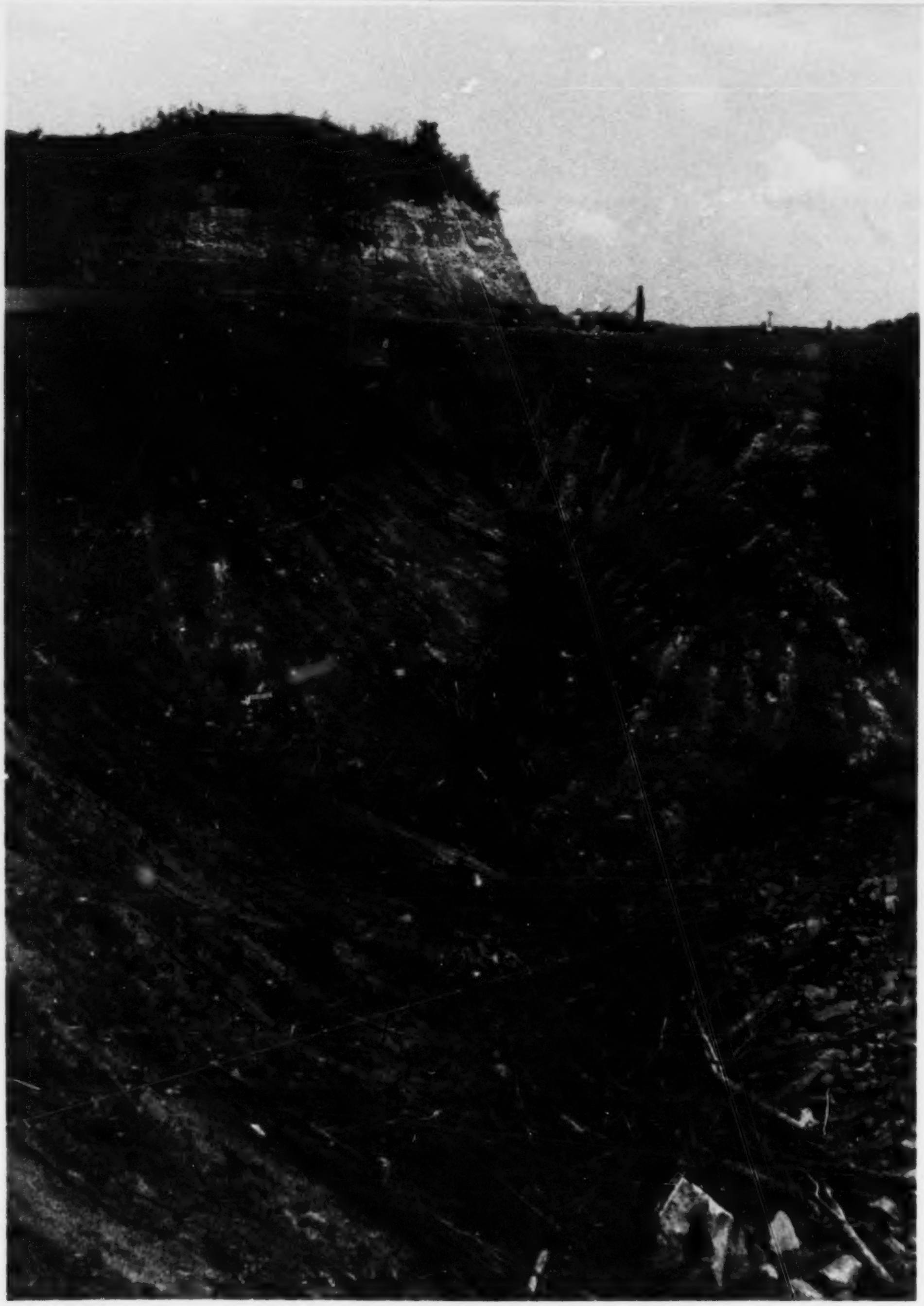

Figure 49. Mountaintop removal method. Note the head-of-hollow in the front being prepared for valley fill.

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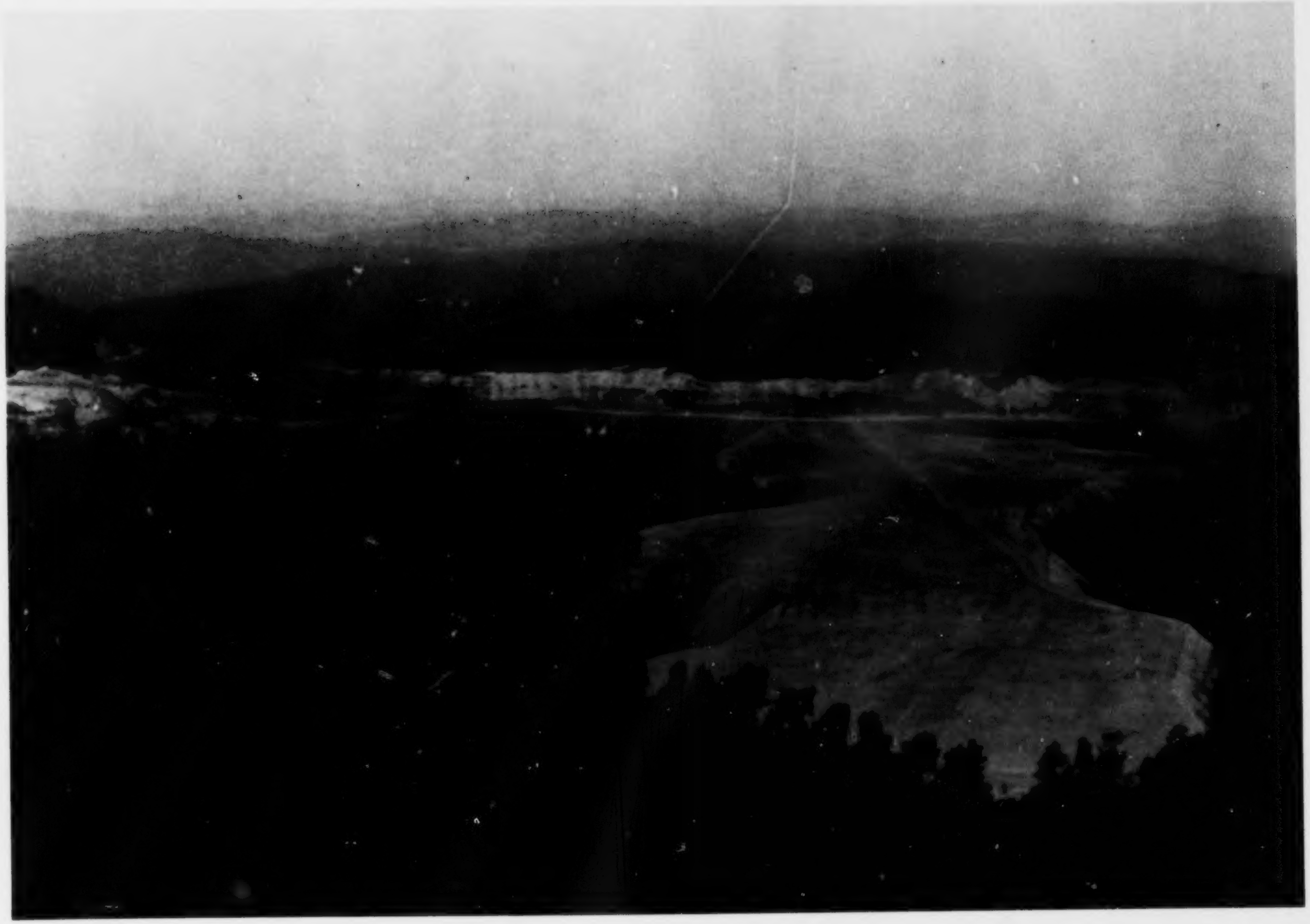

Figure 50. Large area of level ground created as a result of mountaintop removal mining in West Virginia.

Open-Pit Mining

True application of open-pit mining techniques to the exploitation of coal resources is limited to two specific occurrences. In one case, the coal outcrops, but dips so steeply that simple overcasting cannot be easily practiced, due to limitations on equipment reach. The value of the coal is such that the overburden can be stripped in prismatic sections and the pit can be advanced along the strike of the vein. In the other case, the coal is very thick and is overlain by shallow or thick cover. Therefore, benches are required in the coal and the overburden (Figure 51).

While open-pit mining of coal is practiced in the United States, these are in places outside the geographic area of this report.

SPECIAL CONSIDERATIONS - MULTISEAM MINING

Multiple-seam surface mining is done with many types and combinations of equipment. The mining of a second seam of coal often reduces the stripping ratio and therefore provides an overall reduced stripping cost. More selective machines are constantly being designed for increased production and reduced operating costs in special applications, such as removal of both thin partings and coal seams. Flexibility and mobility of this new equipment, combined with lower capital investment, offer great inducement in multiple seam operations or multimine situations.

Mining a second seam of coal and handling the parting in accordance with economic and environmental controls presents a number of problems and poses a challenge to planners. In the context of total resource extraction, all available resources such as valuable claybeds, limestone strata, etc. are best extracted with the extraction of the coal seam under consideration. On the technical side, it is necessary to use a carefully planned sequence of mining and correct combination of available equipment. Loading and hauling units and overburden stripping must be matched.

Porter (1971) conducted a survey of multiple-seam strip mining operations and developed an economic feasibility model. All of Porter's case studies involve stripping shovels, draglines, or tandem operations from the East or Midwest. It is a comprehensive review of conventional multiple-seam (usually two-seam) operations in these regions. Another examination of multiple-seam operations is found in a report by Stefanko et al. (1973), (Figures 52 and 53). Although intended as a preview of all strip mining methods and equipment, the case studies presented cover some multiple-seam mines, particularly in the Midwest and Northern High Plains. Figure 54 shows a photograph of a multiple-seam operation with a 22.93 cubic meter (30 cu-yd) dragline in Pennsylvania.

The ratio of coal thickness to parting thickness is important in the consideration of haulback methods. When the thickness of the parting layer is large compared to the thickness of the coal, space for stoppage of spill is limited and the parting must be hauled upgrade for spoiling. Short,

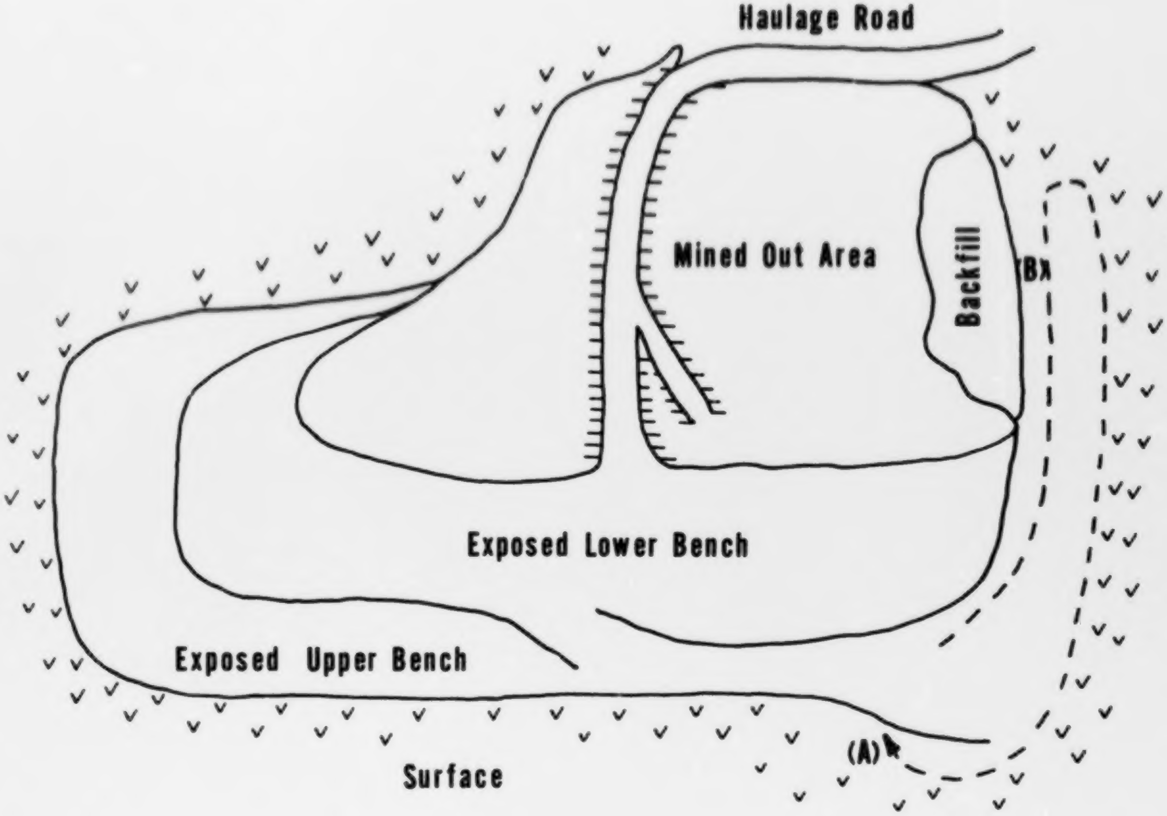
Haulage Road
Mined Out Area
Backfill
(B)
Exposed Lower Bench
Exposed Upper Bench
Surface
(A)

Figure 51. Plan view of an open-pit coal mine.

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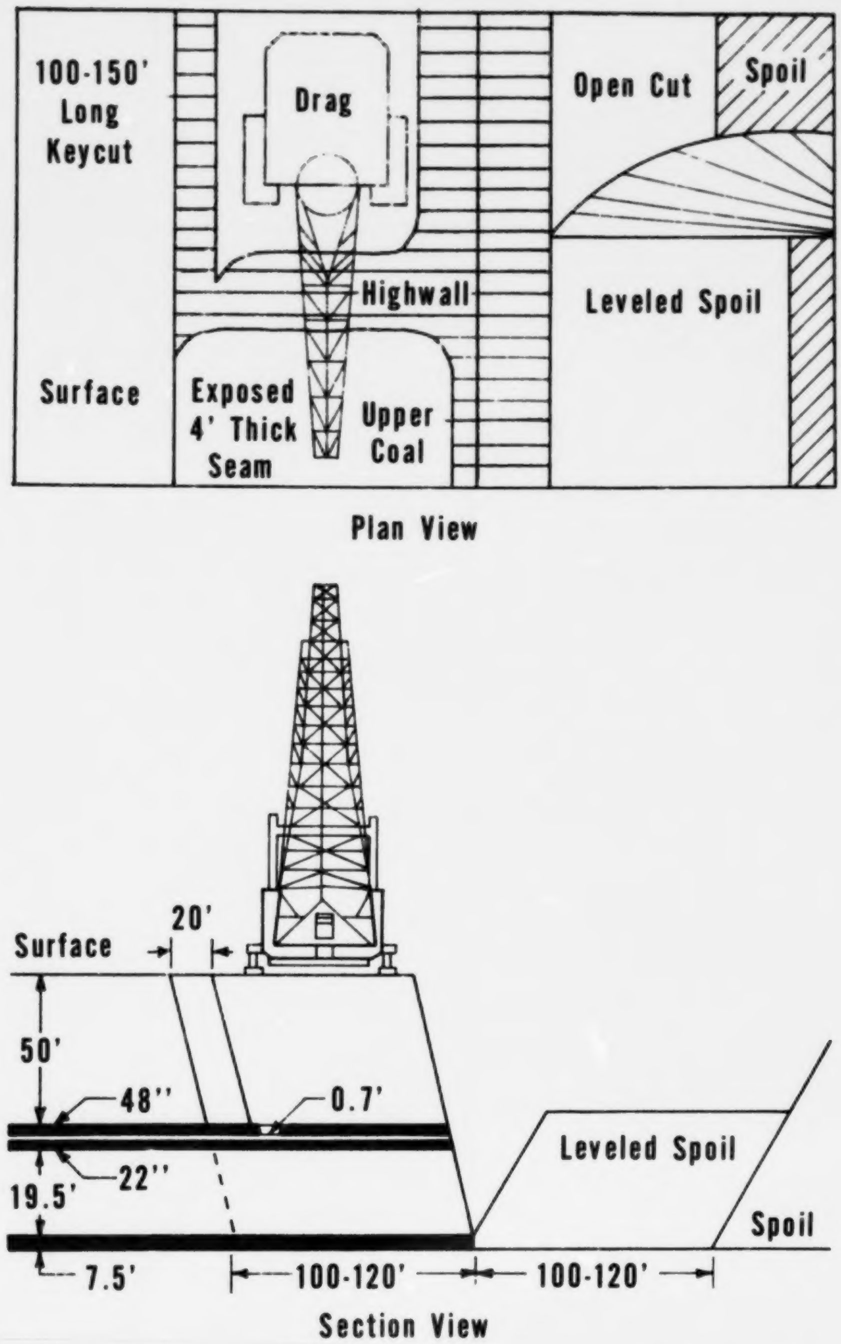
100-150' Long Keycut
Drag
Open Cut
Spoil
Highwall
Leveled Spoil
Surface
Exposed 4' Thick Seam
Upper Coal
Plan View
20'
Surface
50'
48"
0.7'
22"
19.5'
Leveled Spoil
Spoil
7.5'
100-120'
100-120'
Section View

Figure 52. Multiseam stripping operation with a dragline (Stefanko, Ramani, and Ferko, 1973).

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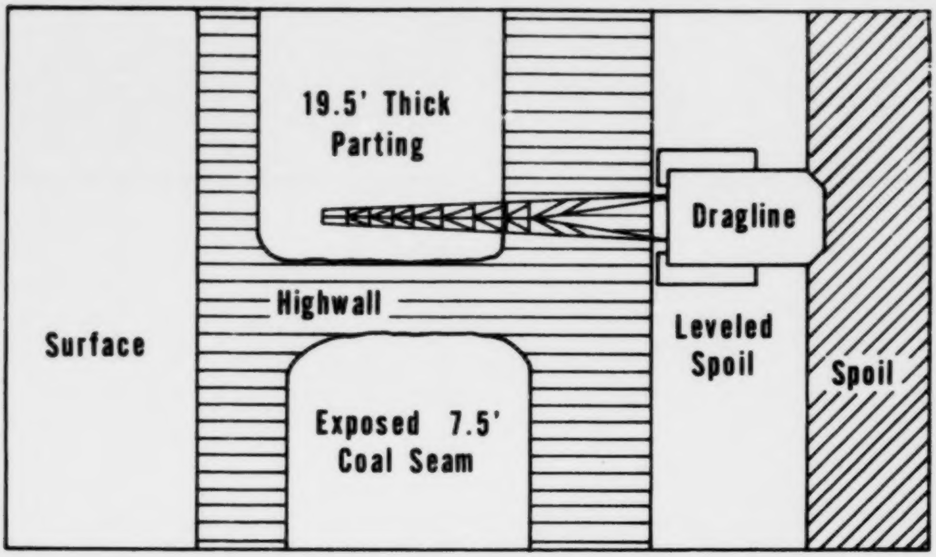
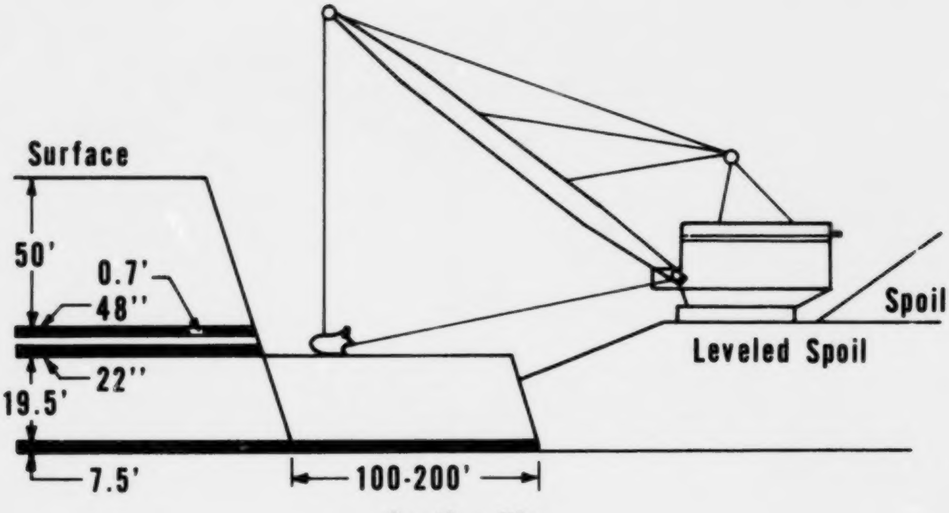

19.5' Thick Parting
Dragline
Highwall
Surface
Leveled Spoil
Spoil
Exposed 7.5' Coal Seam

Plan View


Surface
50'
0.7'
48"
22"
19.5'
7.5'
100-200'
Spoil
Leveled Spoil

Section View

Figure 53. Dragline exposing lowest seam from leveled spoil (Stefanko, Ramani, and Ferko, 1973).

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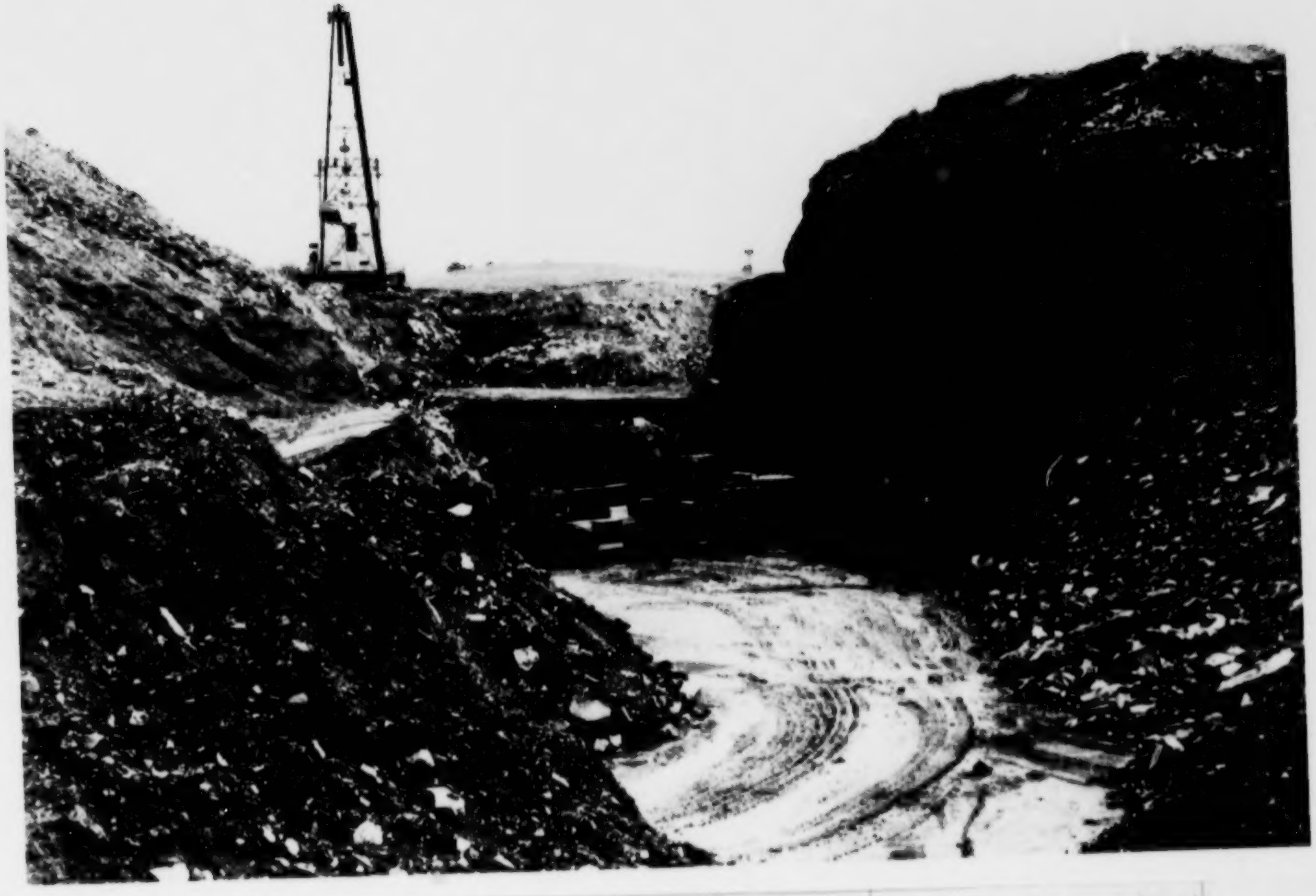

Figure 54. A two-seam operation with a 30-cubic-yard dragline in Pennsylvania.

steep grades encountered with loaded haulage vehicles quickly nullify transfer and stowage benefits.

Porter (1971) details a tandem operation using a stripping shovel and dragline to mine three seams of coal. The operational configuration is shown in Figure 55. A 1050B shovel exposes the first two seams, and a Marion 8800 dragline exposes the bottom seam while simultaneously rehandling 36 percent of the shovel spoil. At intervals, the imbalance between the primary stripping machines necessitates the use of the dragline on the middle parting in order for the 1050B to maintain a proper lead. The rehandle is nonproductive work and the dragline output is about 30 percent less in the shown operating mode than would be expected in a conventional (highwall position) stripping situation. Some of this loss can be attributed to the 180 degree swing required while stripping from the spoil side. Another problem area noted, less related to overburden and parting handling but relevant to multiple-seam operations, is haulage from three levels. Haulage roads are brought into the pit about midway between the upper two seams. Access to the lower seam is gained by an incline dug in the face of the spoil parallel with the pit. The entire operation requires a longer than normal pit to allow maneuvering room. Although the pit is a good operation from a conventional view, the thicker seams of the West offer possibilities for stowing. Stowing would eliminate rehandle, ineffective dragline application, and complex timing sequences as machines pass one another. Effective stowing could shorten the required open-pit length. Although the thinner coal seams will not allow efficient backstowing, the problems encountered in conventional multiple-seam mining are evident.

One multiple-seam operation in Illinois utilizes one shovel to uncover two coal seams. A Marion 5761 shovel strips both seams as two active, parallel cuts. An average of 12.19 meters (40 feet) of top overburden is removed with a 180 degree swing and the seven-foot parting is stripped from directly in front of the shovel with a 90 degree swing. The 1.67 meter (5.5 ft) thick top seam and associated thin parting create this unique situation. The shovel can selectively remove overburden as the top of the parting is only 3.81 meters (12.5 ft) above the top of the second seam on which the shovel sits. The volume of material in the innerburden is minimal and leaves sufficient spoiling room for the top overburden. Although the seams are exposed simultaneously, they are effectively separate slices and the overburdens are shot separately. Utilization of the single unit is good in this particular case. The reach available permits easy handling in a pit 45.72 meter (150 ft) wide. One desirable result is burial of the parting material. Exposure of both seams is another benefit as this allows blending of the coals from the two seams.

Another example of the use of one shovel to mine two seams of coal is near Central City, Kentucky (Figure 56). Although the mining method is not identical to the plan proposed here, it is similar in that one shovel is used to excavate two separate strata of overburden material simultaneously.

The stripping shovel, a Marion 5761 with a 49.69 cu m (65 cu yd) dipper, operates on top of the lower coal seam to remove the overburden and the parting layer. Overburden thicknesses average 12.19 meters (40 ft) and the

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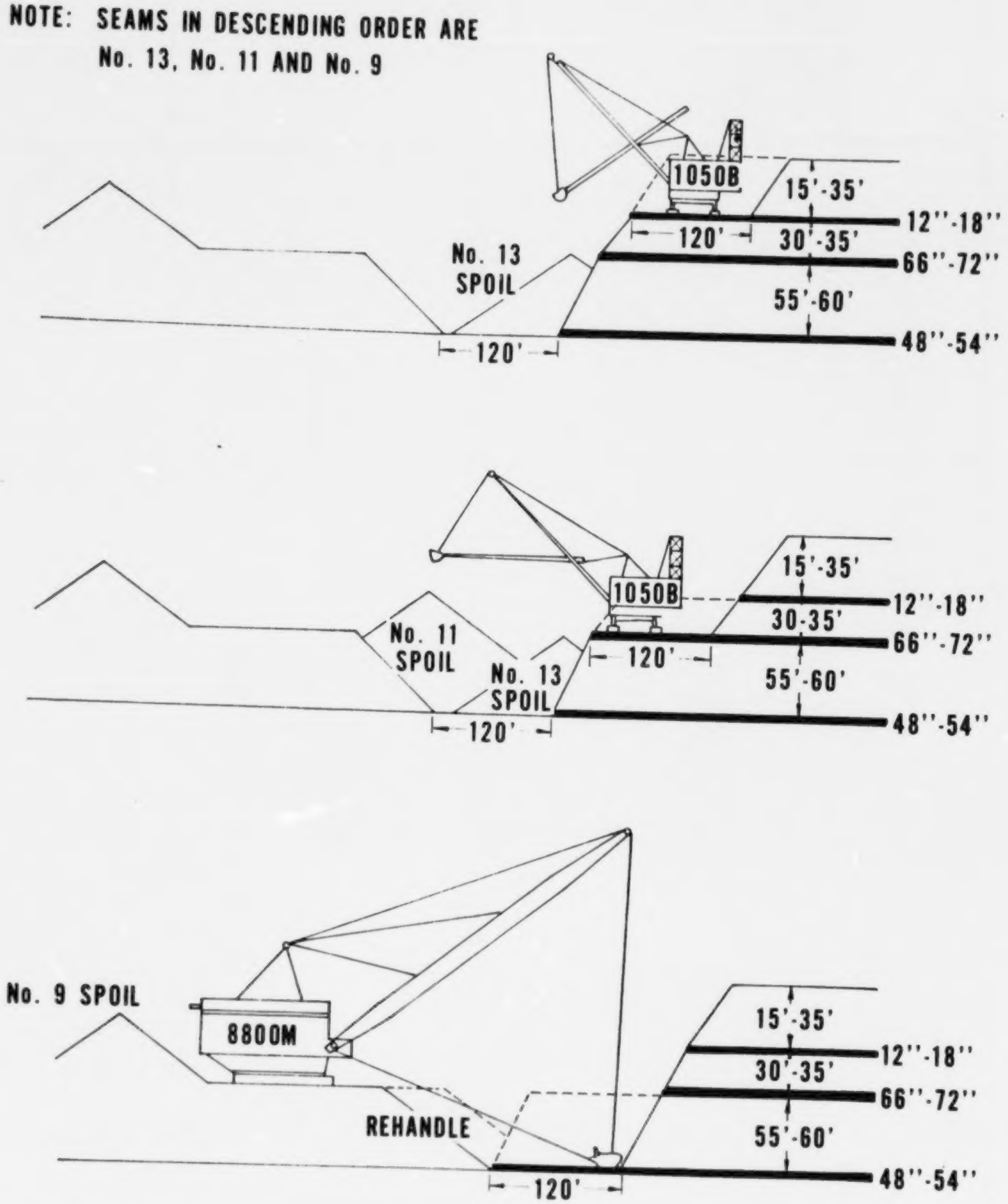
NOTE: SEAMS IN DESCENDING ORDER ARE
No. 13, No. 11 AND No. 9
1050B
15'-35'
12"-18"
120'
30'-35'
66"-72"
No. 13 SPOIL
55'-60'
48"-54"
120'
1050B
15'-35'
12"-18"
30-35'
66"-72"
No. 11 SPOIL
No. 13 SPOIL
120'
55'-60'
48"-54"
120'
No. 9 SPOIL
8800M
15'-35'
12"-18"
30'-35'
66"-72"
REHANDLE
55'-60'
48"-54"
120'

Figure 55. Shovel-dragline tandem operation for multiseam mining (Porter, 1971).

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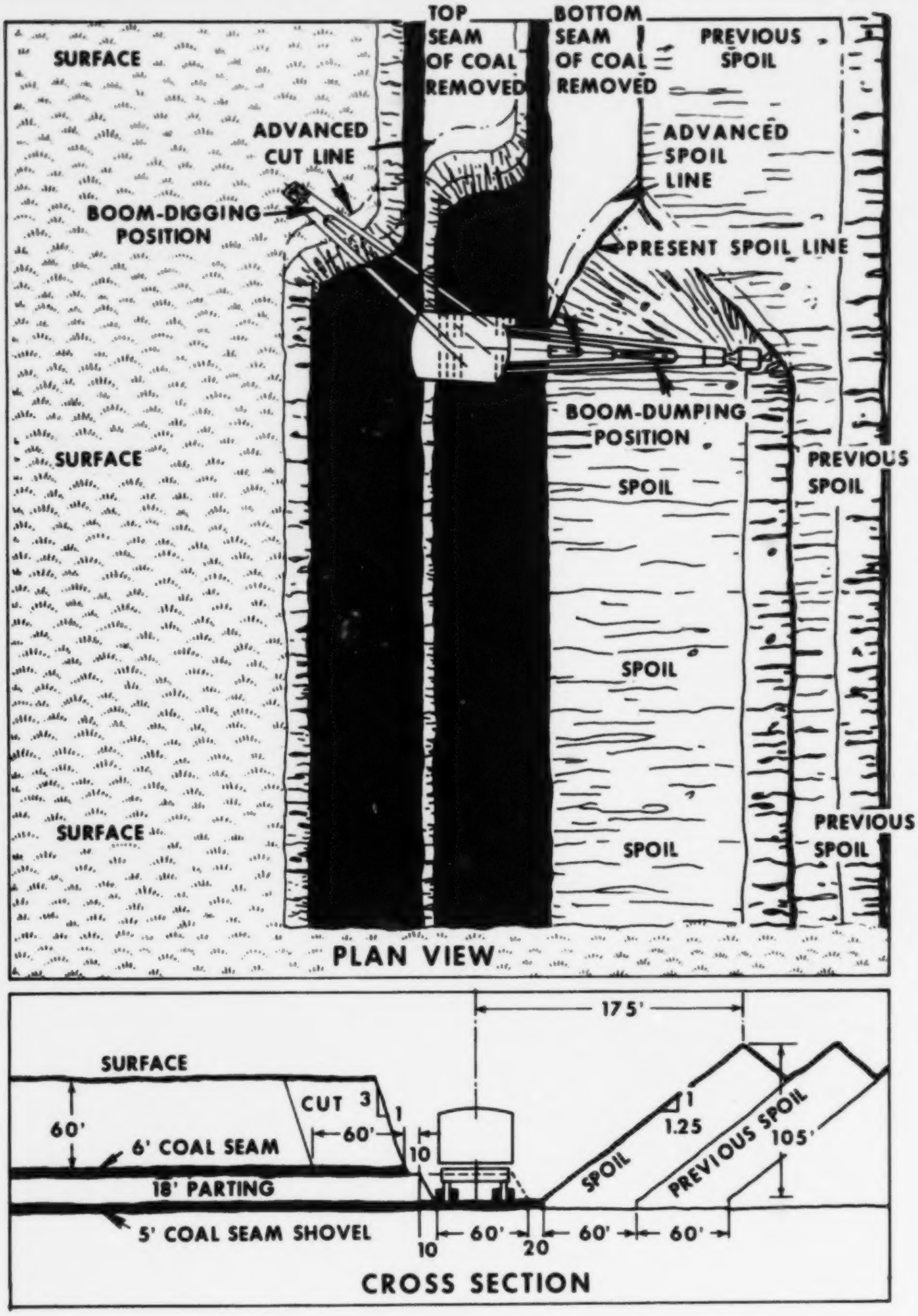
TOP SEAM OF COAL REMOVED
BOTTOM SEAM OF COAL REMOVED
PREVIOUS SPOIL
SURFACE
ADVANCED CUT LINE
ADVANCED SPOIL LINE
BOOM-DIGGING POSITION
PRESENT SPOIL LINE
BOOM-DUMPING POSITION
SURFACE
PREVIOUS SPOIL
SPOIL
SPOIL
SURFACE
SPOIL
PREVIOUS SPOIL
PLAN VIEW
175'
SURFACE
CUT
3
1
60'
60'
6' COAL SEAM
10
1
1.25
105'
18' PARTING
SPOIL
PREVIOUS SPOIL
5' COAL SEAM SHOVEL
10
60'
20
60'
60'
CROSS SECTION

Figure 56. Multiseam stripping operation with a shovel.

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parting layer averages 2.13 meters (7 ft) in thickness. Two working faces in a double-wide cut are advanced at the same time. The parting layer is excavated from in front of the shovel and dumped into the empty cut. The 5761 with a 57.91 meter (190 ft) boom then digs to the highwall side to remove the overburden which is also spoiled into the empty cut and on top of the parting. This type of operation requires a large shovel with good reach, but the stowing and selective mining concept indicates that such a shovel application is feasible. A variable pitch dipper would increase the efficiency of such a mining method.

Grim and Hill (1974) outline three methods for multiple-seam mining which are applicable in steep terrains:

Method No. 1--If the overburden from the upper seam will not reach the bench of the lower seam, treat each seam as a separate mining operation, mining the lower seam first. This bench may be used to store spoil produced during stripping of the upper seam.

Method No. 2--If the overburden from the upper seam will reach the bench of the lower seam, mine the lower seam in advance of the seam above. Grading should be delayed on the lower bench in order to catch big rocks from the upper seam and bury them in the pit. In no instance can spoil from the upper seam extend more than one-half the distance from the highwall to the edge of the solid bench of the lower seam.

Method No. 3--If both seams appear in the same highwall, separated by more than 7.6 meters (25 feet), and two or more cuts are planned, the coal should be recovered from the bottom seam first. If the seams are separated by less than 7.6 meters (25 feet), mine from the upper seam down, recovering both seams in one systematic operation (Figure 57, Steps 1 and 2). Lateral movement of the spoil is recommended.

SUMMARY

This chapter has briefly reviewed the more commonly used surface coal mining methods. Several examples were presented on specific equipment applications, including multiseam mining operations. While haulback methods are becoming popular in the hilly terrains of the east, area mining with simple overcasting is most popular in the interior region. Tandem operation of large strip equipment (bucket-wheel excavator, shovels. draglines) is quite common in Illinois. The concern for concurrent land reclamation, and topsoil replacement has seen the introduction of a fleet of wheeled vehicles for topsoil and overburden removal.

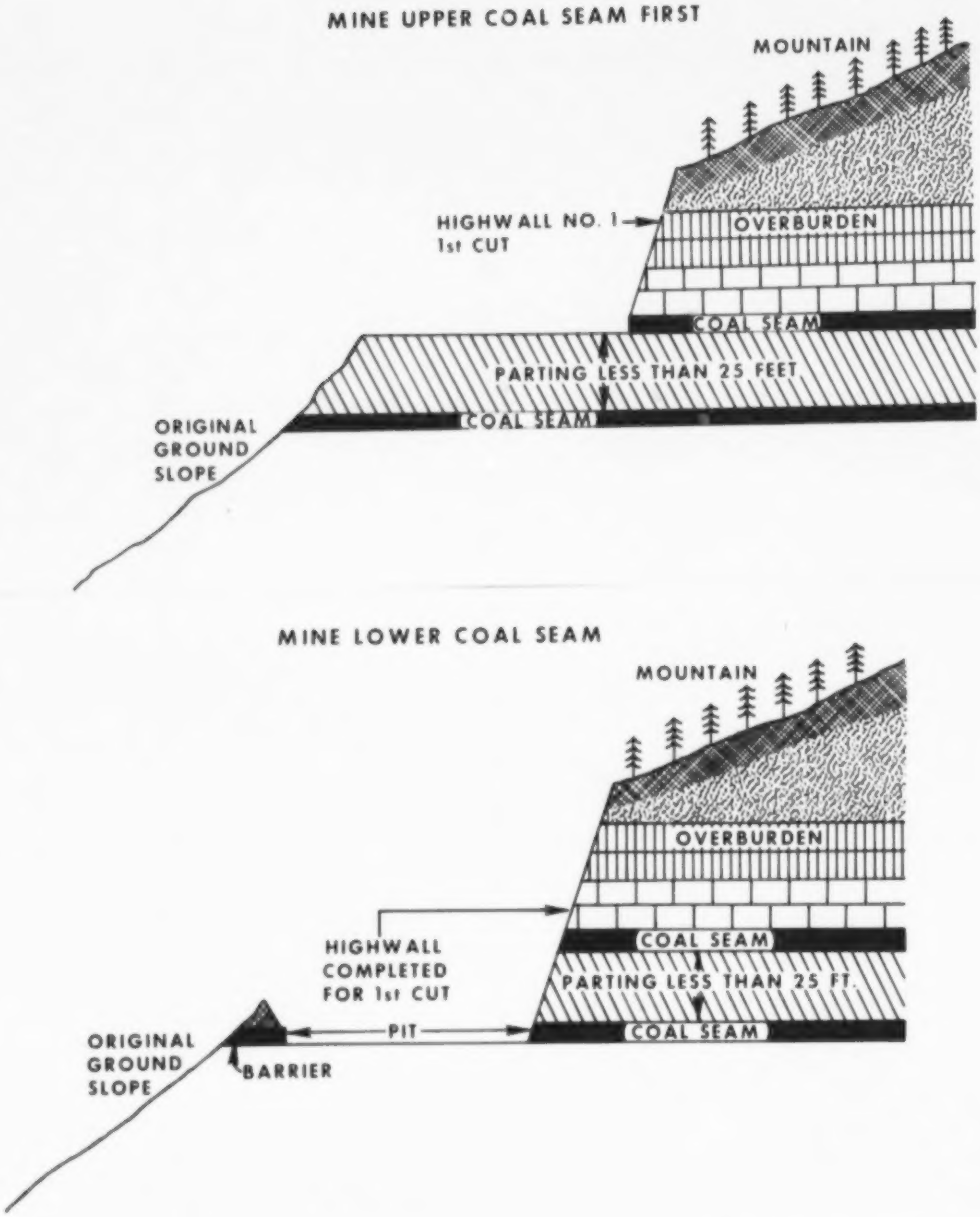
MINE UPPER COAL SEAM FIRST
MOUNTAIN
HIGHWALL NO. 1
1st CUT
OVERBURDEN
COAL SEAM
PARTING LESS THAN 25 FEET
COAL SEAM
ORIGINAL GROUND SLOPE
MINE LOWER COAL SEAM
MOUNTAIN
OVERBURDEN
HIGHWALL COMPLETED FOR 1st CUT
COAL SEAM
PARTING LESS THAN 25 FT.
PIT
COAL SEAM
ORIGINAL GROUND SLOPE
BARRIER

Figure 57. Multiple-seam mining method: two seams less than 25 feet apart in the same highwall.

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SECTION 5

FRAGMENTATION PRACTICES

INTRODUCTION

Fragmentation encompasses several different unit operations in the surface mining of coal. These operations combine to provide for the fracturing of both the overburden and coal, thereby simplifying the subsequent materials handling procedures. In some favorable conditions, neither the overburden nor the coal may exhibit such strength as to require fragmentation prior to actual excavation. In the most restrictive case, the overburden and the coal may both be so strong as to require noncurrent blasting. As is typical in the mining industry, composite situations exist whereby portions of the overburden or coal can be directly excavated while other portions must be prepared prior to the digging operation.

As far as overburden preparation is concerned, a range of possibilities is presented in the fragmentation unit operations. Overburden fragmentation and preparation may be accomplished in some situations by ripping with bulldozers. As an intermediate case, widely spaced drilling and blasting may be sufficient. In the limiting case, a regular and intensive drilling and blasting approach may be required. Where blasting is required, various considerations can affect the drilling layout. The size and nature of the drillholes may vary, thus affecting spacing. Further, the proper type of explosive, either high or low strength, must be carefully examined.

Although the stated purpose for rock fragmentation is to fracture the overburden and coal for loading, there are other secondary, but vital, purposes of a properly designed fragmentation process. These purposes include product size, and rotation and translation.

The size of the broken materials should be small enough to fit into the bucket of the excavating machine and to eliminate secondary blasting, yet small particle generation should be kept to a minimum for dust control. While the largest size is clearly dictated by the dimensions of the machinery, the lower limit is rather difficult to determine. Suffice it to say that fragmenting to a size smaller than that required is wasteful in explosives and cost.

Rotation and translation (throw) refers to the manner in which the blasted material comes to rest. If the explosive force achieved is too great, then the material may move beyond the affected area, thus causing difficulty in loading. Further, a properly designed blasting layout can rotate the fragments to improve their loadability characteristics.

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Three additional factors in fragmentation relate to the precautions needed to minimize the environmental effects of blasting. The blast design must minimize the environmental effects of blasting. The blast design must minimize fines, noise and vibration. Critical to the environmental effects is fines production (dust). Over-designing the blasting rounds can, in addition to fines production, cause conditions of excessive noise and vibrations. If not properly controlled, the above-mentioned three problems can have as great an impact on fragmentation practices as the production requirements.

The parameters on which the design of fragmentation processes are based relate to two general areas. One is the previously mentioned total operation design, i.e., the dimensions of the area to be prepared, the desired fragmentation, and the economic considerations. The second area concerns the properties of the material to be broken, i.e., primarily the strength of the material and its response when subjected to high energy impulses.

A solid mass exhibits different responses to the application of energy. Basically, the response differs with the absolute quantity of energy introduced and the time over which the energy is applied. Rocks can absorb small impulses of energy, or even large quantities of energy applied evenly over a long period of time, and can deform without collapse. Therefore, the goal in breaking rock is to apply a large enough force in a short enough time to create fractures in the mass and to rotate and translate the resultant fractured mass.

The simplest case of fragmentation is that in which no explicit fragmentation process is required, or the case where the excavator (bucketwheel, dragline, shovel, loader, scraper, or bulldozer) can dig the material without prior treatment. In this case, the digging machine is capable of delivering an excess of the total amount of energy that the mass can withstand in the static case. The next extension of the simple case is that in which the excavator is just barely incapable of digging the material. In this case, it is possible to concentrate energy through use of a ripper. A ripper concentrates the tractive effort of the dozer (which can be considerable) at one point, the ripper tooth. The applicability of direct ripping or direct excavation without fragmentation is determined in the specifics of equipment selection for either coal or overburden handling. In many cases, small operators may choose to use such techniques at marginal conditions in an effort to save cost in drilling equipment and personnel. Large operators may choose to use direct excavation where mobility is a key to successful operation. In the Eastern United States, drilling and blasting are necessary prior to excavation in most operations.

DRILLING

Where the strength of overburden material is such that blasting is required prior to excavation, drilling precedes the use of explosives. Quite simply, drilling is the unit operation which produces the hole for the

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placement of explosives. Drilling, for the mining of coal by surface techniques, is categorized as rock penetration by mechanical attack. Mechanical attack is further subdivided into two methods - percussive and rotary action.

Beyond the mechanics of attack, drilling is also categorized by the manner of application. Specifically, the drilling of overburden for the surface mining of coal can be approached in three manners:

1. Vertical - conducted from the top of the highwall down to the coal perpendicular to the strata bedding planes.
2. Horizontal - conducted from the pit floor parallel to the bedding planes of the strata.
3. Inclined - since the shortest drilling distance to a pitching seam is perpendicular to strata bedding planes, inclined drilling is conducted from the top of the highwall normal to the coal seam's angle of pitch.

Since most coal seams in the East are tabular, the third option is rarely encountered.

PERCUSSION DRILLING

Penetration of overburden by percussion drilling is affected by successive impacts of the bit into the rock. The bit is rotated between blows to aid in the crushing and chipping process. Percussion drilling is used when small diameter holes must be driven through hard rock where rotary units are difficult to employ.

There are a number of advantages of percussion drilling. First, small diameter holes can be drilled through hard rock. The rigs themselves are light, less expensive, and are quite maneuverable. Finally, one man can operate the rig through a wide range of drilling angles (Morrell and Unger, 1973).

Percussive drills are subdivided into two categories, surface-mounted and down-the-hole. The percussive action of surface-mounted units is transmitted to the bit by a prime mover which remains outside the hole. In comparison, the prime mover for a down-the-hole drill actually strikes the bit and follows it into the drilled holes.

The most widely employed percussive drills for surface mining are surface mounted. Pneumatically powered, these drills also use the air for flushing the cuttings from the hole. Since the addition of drill steel can decrease the available drilling energy nearly 50 percent, surface-mounted percussive drills are used primarily where the drilling depth is less than 30.48 meter (100 ft) (Morrell and Unger, 1973).

Since the bit must be rotated between blows to effectively chisel the rock, surface-mounted percussive drills are either rifle-bar or independently rotated (Table 20). The drill steel in a rifle-bar rotation unit is

TABLE 20. SPECIFICATIONS OF SOME RIFLE-BAR ROTATION AND INDEPENDENT ROTATION PERCUSSIVE DRILLS (AFTER MORRELL AND UNGER, 1973)

Specifications	Rifle-Bar Rotation			Independent Rotation		
Hole size, in.	$1\frac{1}{2} - 2\frac{1}{2}$	$2 - 3\frac{1}{2}$	$3\frac{1}{2} - 5\frac{1}{2}$	$1\frac{1}{2} - 2\frac{1}{2}$	$2\frac{1}{2} - 4$	$3\frac{1}{2} - 5\frac{1}{2}$
Piston diameter, in.	$3\frac{1}{2}$	$4\frac{3}{16}$	$5\frac{1}{2}$	$3\frac{1}{2}$	$4\frac{1}{2}$	$5\frac{1}{2}$
Drill weight, lb	112	155	385	---	290	690
Drill length, in.	30	29	40	---	39	55
Blow frequency,[1] bpm	1,760	1,975	1,616	1,970	1,640	1,250
Energy per blow,[1] ft-lb	135	160	275	117	228	375
Energy output,[1] ft-lb per min.	238,000	316,000	444,000	230,000	374,000	469,000
Air consumption,[1] cfm	195	340	900[3]	325[2]	560[3]	1,050[3]

[1] At 100-psi operating pressure. [2] Includes air motor. [3] Includes blow air for flushing.

automatically rotated by a fluted rifle bar and nut on the block stroke. Since rotation is automatic, the ability to vary the rotational speed is minimal. Independently rotated drills are, as the name implies, rotated by an independent motor. This adds flexibility to the system when rock of varying strength is encountered. For example, harder rock requires a decrease in rotational speed and an increase in the impacting force. Rotational speeds of 200 to 400 rpm are common for 50.8 mm (2 inch) bits, while 152.4 mm (6 inch) bits are usually operated at less than 100 rpm.

Surface-mounted drills offer the following advantages (Morrell and Unger, 1973):

1. Surface-mounted drills can be more powerful and can penetrate faster in shallow holes than down-the-hole drills because, unlike the latter, the drill-piston diameter can be greater than the hole diameter.
2. The minimum hole diameter for a down-the-hole drill is 101.60 mm (4 in), whereas a surface-mounted drill is capable of even smaller diameters.
3. Unlike a down-the-hole drill, the loss of the drill unit due to hole caving is not a problem for surface-mounted units.

However, surface-mounted drills have the following disadvantages:

1. Total penetration depth is limited to 30.48 meters (100 ft) because the loss of drilling energy, due to the addition of five joints of drill steel, can approach 50 percent.
2. Holes greater than 152.4 mm (6 in) in diameter are not common.
3. Unlike down-the-hole drills, noise is not facilitated by hole penetration.
4. Drill rod life is reduced due to the continued transmission of impaction energy.

Down-the-hole percussive drills are employed where drillholes greater than 30.48 m (100 ft) are required and hole diameters of up to 304.8 mm (12 in) are preferred (Table 21). Since the percussive unit follows the bit into the hole, a clearance of approximately 12.7 mm (0.5 in) must be provided by the hole diameter for the passage of cuttings around the drill unit. The rotational motor is the only part of the operation that takes place out of the hole. A typical rotation range is 10 to 100 rpm (Morrell and Unger, 1973).

Down-the-hole drills have the following advantages:

1. Penetration rates are maintained because the drill-to-bit distance is constant, unlike that of a surface-mounted drill, and percussive energy is not lost.
2. The drill noise is dampened as soon as the unit enters the hole.
3. Since they do not transmit impact energy, drill rods have a longer life than those employed with surface-mounted drills.

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TABLE 21. DOWN-THE-HOLE DRILL SPECIFICATIONS
(AFTER MORRELL AND UNGER, 1973)

	Hole Diameter		
Drill Specifications	4 In.	6 In.	8 In.
Overall diameter, in............	$3\frac{1}{4}$	$5\frac{1}{4}$	$6\frac{1}{8}$
Tool weight, lb.................	90-100	180-200	300-350
Tool length, in.................	40-45	40-60	50-65
Piston diameter, in.............	$2\frac{1}{2}$	$4\frac{1}{2}$	5
Operating pressure, psi.........	100-250	100-250	100-250
Air consumption, cfm............	100-500	300-600	500-1,600

Down-the-hole drilling has the following disadvantages:

1. Since the minimum hole diameter is dictated by the drill diameter, the minimum hole size is four inches.
2. Hole caving can make drill recovery difficult, if not impossible.
3. Since larger bit diameters are required for each drill size, the amount of energy that can be applied to a bit is limited.

Percussive drill bits are divided into two categories for surface mining applications, cross bits and button bits. The tungsten carbide cross bits are either of an "X" (at an angle nearly 90 degrees) or a cross (90°) configuration. Button bits, in sizes from 50.8 to 165.1 mm (2 to 6.5 in), are flat-bottomed with cylindrical tungsten-carbide inserts. Cross bits can usually be sharpened 20 times, while button bits do not require sharpening.

ROTARY DRILLING

Rotary drilling, the most versatile form of overburden penetration for surface mining of coal, is conducted by rotating a rigid, tubular string of rods to which is attached a rock-cutting bit. Thus, the drilling energy is supplied by rotation as well as thrust. The cutting action consists of either abrasion, scraping, spalling, or chipping, as well as a combination of the above. The size range for rotary-drilled blasthole diameters is from 101.6 mm (4 in) to 381 mm (15 in) with 228.6 mm (9 in) and 304.8 mm (12 in) being common sizes.

For the surface mining of coal, most rotary drilling rigs are truck-mounted, although crawler-mounted units are also used. All drilling rigs are equipped with jacks for leveling on uneven ground. The power source for the rotary drill rigs is either electric (for economy) or diesel (for flexibility in tramming).

Drill steel, of up to 15.24 m (50 ft) in length, is stored on a circular rod carrier which rotates in a mast for the automatic changing of drill steel. Although most drilling is vertical, the mast can be tilted up to 30 degrees off vertical for angled drilling.

There are two types of bits that are used for rotary drilling, drag or rolling cutter. Drag bits are applicable in soft to medium-soft shales or their equivalents. Since their effectiveness is dependent upon a successful scraping action, drilling thrust is lower for drag bits than for rolling cutter bits, but the opposite is true for torque requirements. The bits are usually fitted with replaceable tungsten-carbide cutters.

Rolling cutter bits, though available in the two-cone configuration, are usually of the three-cone style (Table 22). The rolling cones have either large, steel teeth (for drilling through soft formations) or small, tungsten-carbide inserts (for drilling through hard formations). Table 23 relates bit size and weight to the strength of the rock.

TABLE 22. THREE-CONE ROLLING CUTTER BIT DIMENSIONS AND HOLE VOLUME PRODUCED (AFTER MORRELL AND UNGER, 1973)

Bit Diameter, In.	Pin Size API* Reg., In.	Bit Weight, Lb	Volume of Hole Drilled, Cu Ft per Ft
$5\frac{5}{8}$	$3\frac{1}{2}$	26	0.173
$6\frac{3}{4}$	$3\frac{1}{2}$	35	0.249
$7\frac{3}{8}$	$3\frac{1}{2}$	49	0.279
$7\frac{7}{8}$	$3\frac{1}{2}$	60	0.338
$8\frac{1}{2}$	$4\frac{1}{2}$	73	0.394
$8\frac{3}{4}$	$4\frac{1}{2}$	75	0.418
9	$4\frac{1}{2}$	77	0.442
$9\frac{7}{8}$	$6\frac{5}{8}$	122	0.532
$10\frac{5}{8}$	$6\frac{5}{8}$	141	0.616
$12\frac{1}{4}$	$6\frac{5}{8}$	200	0.818
15	$6\frac{5}{8}$	306	1.227

* American Petroleum Institute.

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TABLE 23. BIT WEIGHT AS A FUNCTION OF ROCK STRENGTH AND BIT SIZE (AFTER MORRELL AND UNGER, 1973)

Rock Strength	Weight per In. of Diameter, Lb	Recommended Bit Loading for Bit Size Indicated, Lb		
		4-6 In.	6-9 In.	9-12 In.
Very soft formations: overburden, soft shales, limestone and evaporites	1,000-4,000	4,000-24,000	6,000-36,000	9,000-48,000
Medium formations: limestone, dolomites and sandstones	3,000-5,000	12,000-30,000	18,000-45,000	27,000-60,000
Hard and very hard formations: basalt, granit, quartzite, taconite	4,000-8,000	16,000-40,000*	24,000-60,000*	36,000-100,000

* Maximum weight for largest bit size indicated.

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The major advantage of rotary drilling is its rapid drilling rate. This is due to the fact that, unlike percussion drilling, rock penetration is uniform with depth (Morrell and Unger, 1973).

VERTICAL AND HORIZONTAL DRILLING

There are basically two approaches to drilling overburden for the surface mining of coal, vertical and horizontal drilling. Although vertical drilling appears to be the overwhelming choice for surface mining applications, the horizontal method can be used effectively under certain conditions.

Vertical drilling consists of drilling from the top of the highwall perpendicular to the coal seam. This makes the use of bulk explosives possible. It is also the preferred approach where overburden, which consists of competent strata, exceeds 15.24m (50 ft) in depth or massive stone occurs at an elevation higher than that which can be reached by horizontal drilling. Because common practice in vertical drilling is to penetrate to the coal and then backfill approximately four feet, there is less shattering of coal with this approach.

Horizontal drilling, a relatively new development for the surface mining of coal, is usually conducted by a twin-mast rotary drill. Both holes, which can be drilled either simultaneously or independently, can be varied from 8.53 m (28 ft) to 11.58 m (38 ft) centers and can be drilled from 10 degree above to 12 degree below the horizontal. Typical hole diameters vary from 171.45 mm (6-3/4 inches) to 269.87 mm (10-5/8 inches).

Where the only portion of the overburden that must be blasted is located directly above or near the seam, horizontal drilling is applicable since less footage is usually required for the same pit width. Horizontal drills tram on the coal floor which, by eliminating the surface preparation required for vertical drills, allows for faster rates. Although applications are usually limited to a maximum of 30.48 m (100 ft) of overburden, 45.72 m (150 ft) deep holes (approximately two pit widths) can be conveniently drilled at one time (Nordness, 1976).

BLASTING

Effective bank preparation not only deals with the fragmentation of overburden, but also with the minimization of incurred costs. As such, the target for proper overburden preparation is efficiency with regard to the number of holes drilled and the amount of explosives utilized.

It has already been stated that operators must continually evaluate their fragmentation procedures. The amount of explosives employed must be adequate for fracturing the rock to a size which is acceptable for equipment removal. Since the character of the overburden in a large surface mine can change over a wide area, economics alone is a sufficient reason for the emphasis on studying and selecting good fragmentation practices.

SYSTEM COMPONENTS

Blasting systems are constructed of the following components: 1) detonators, 2) primers and boosters, and 3) explosives.

Detonators are devices for producing an explosive decomposition in a high-explosive charge. Initiation of the detonator is by either a safety fuse or by electricity. Detonators consist of two styles, blasting caps or detonating cord.

The most commonly used detonator is the electric blasting cap. It consists of a copper shell, closed at one end, which contains a detonating compound. Because it is placed in the explosive charge, the ignition of the cap, by a current or spark, sets off the entire reaction. The advantages of electric blasting caps are as follows (Dick, 1973): 1) can be safely handled, 2) accuracy of delay periods, and 3) controlled firing time. The following are hazardous conditions which dictate caution when using electric blasting caps (Dick, 1973): 1) lightning, 2) radio frequency energy, and 3) mixing of different brands of blasting caps.

Blasting caps can also be activated with a safety fuse. A safety fuse consists of a potassium-nitrate black powder core enclosed in a covering of textile and waterproofing compound. The fuse is inserted into the open end of the cap, which is then attached to the charge. Burn rates for fuses vary between 98.42 and 131.23 second per meter. Care must be taken when using a safety fuse because of the following conditions (Dick, 1973):

1. Since the fuse cap is open at an end, exposing the explosives, care must be taken in handling.
2. Altitude changes affect burning rates.

However, the following are advantages for using safety fuses:

1. Elimination of the hazard of stray electricity.
2. No need for a power source or associated lead wires.

The detonating cord is a fuse, consisting of a high-explosive core of pentaerythritol tetranitrate (PETN), enclosed in tape, and wrapped with textile yarns, that fires the charge without the assistance of any other detonator between it and the charge. However, an electric blasting cap is needed to start the reaction in the detonating cord. Although blasting caps with fuses can be used, electric blasting caps are preferred because of their ability to control the instant of firing. In fact, the use of two caps at each initiation point is common to guarantee proper firing. Since a detonating cord is insensitive to shock, friction, and lightning, it is quite applicable as a safe and convenient trunkline for firing blasts.

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Initially, applications of detonating cord were instantaneously fired by means of a trunkline. To achieve a delay, electric caps were attached in each hole. Millisecond (M.S.) connectors were then developed to provide delay firing by surface, rather than downline, initiation.

With instantaneous firing, the layout must be designed with attention to the direction of the blast movement. For example, the charges nearest the initiation point should not "cut off" the firing in later holes due to ground movement. Because of this, the trunkline should run parallel to a free face and should also have an open end (Figure 58). With this configuration, detonation should be from the open end. By looping the trunkline ("double trunkline"), a second means of detonating the holes is available if a break occurs in the main trunk. In multiple row blasts (Figures 59 and 60), availability of secondary detonation is provided by cross ties, spaced on intervals of less than 60.96 m (200 ft).

Delay firing, resulting in reduced vibration, improved fragmentation, and less backbreak, is considered to be more desirable than instantaneous firing. Delay intervals where detonating cord is used vary between 5 and 35 milliseconds. For surface mining applications where large-diameter boreholes are spaced on approximately 9.14 m (30-foot) centers, 25-millisecond intervals are used to avoid cut-offs. In fact, to initiate detonating cord downline with millisecond delays attached at the surface, the minimum available interval is 25 milliseconds.

Millisecond delays were quite widely employed prior to the introduction of millisecond connectors. These short-interval delays are attached, on the surface, to each detonating cord downline at the individual holes or to groups of holes by means of a trunkline. With M.S. delays and the absence of a trunkline, noise is reduced; however, precautions must be taken because of the large amount of electrical wiring required.

Millisecond connectors are tied directly into the detonating cord trunklines. Unlike M.S. delays, M.S. connectors come in a variety of delay intervals, 5, 9, 17, and 25 milliseconds. Figures 61 through 65 show sample configurations with M.S. connector placement marked by an "X". Special attention should be drawn to Figure 65 because it shows a configuration for horizontal drilling.

A primer is usually the combination of an explosive cartridge and a detonating cap. By detonation, it sets off the remainder of the charge. The cartridge is placed at one end of the charge with the embedded detonator pointing towards the charge, not away from it. Detonating cord can also be used as a primer. The following precautions should be taken to assure the proper application of primers (DuPont Co., 1967):

1. The blasting cap should be securely inserted into the primer cartridge (deeply into the center) to guarantee that it cannot be pulled out.
2. The wires of electric firing devices or fuses should not be subject to harmful strains.

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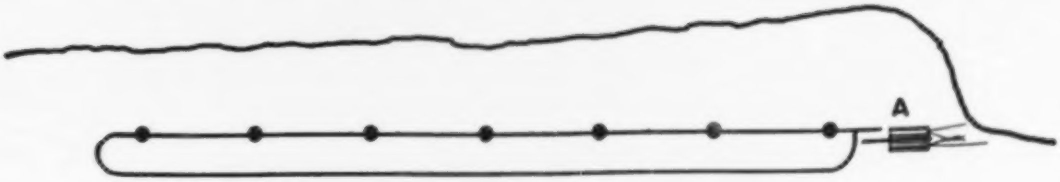

Figure 58. Single row of holes connected by "primacord" and detonated by electric blasting caps or caps and fuse attached to "primacord" at A. Also, note that the "primacord" loop is tied to the trunkline with a right angle connection (After DuPont Co., 1967).

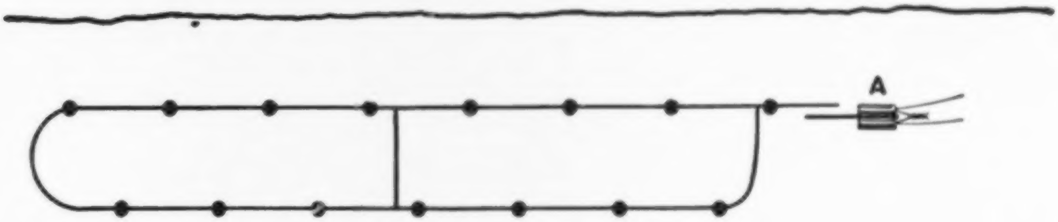

Figure 59. Double row of holes connected by "primacord" (After DuPont Co., 1967).

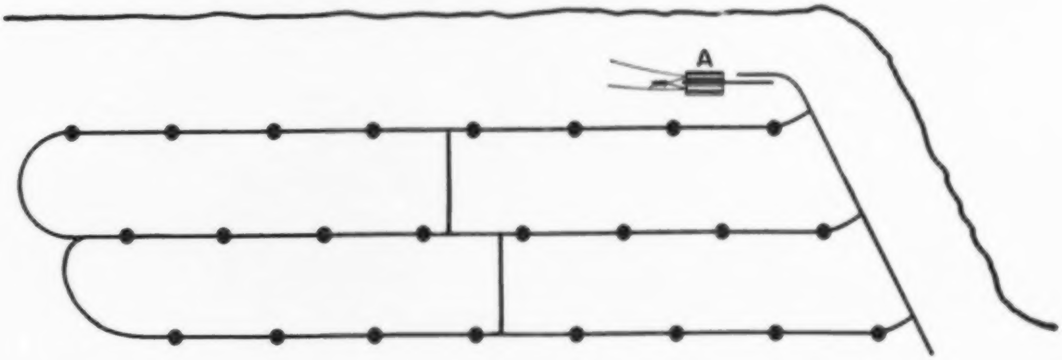

Figure 60. Three rows of holes connected by "primacord" (After DuPont Co., 1967).

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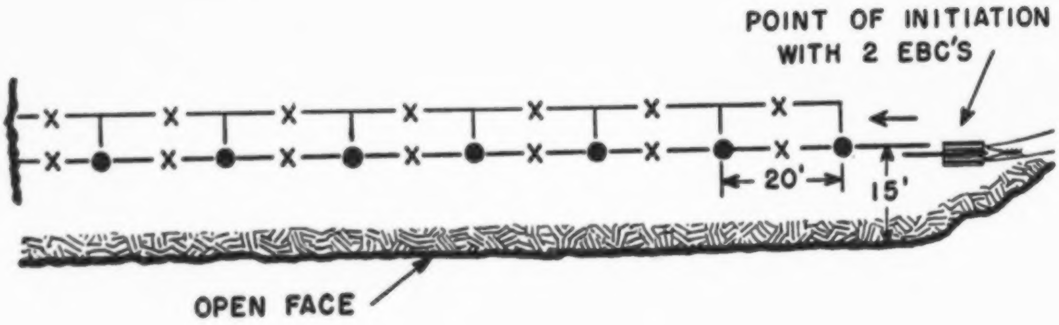
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X
20'
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OPEN FACE

Figure 61. Single row shot fired in sequence."Primacord" M.S. connectors inserted at points "X" (After DuPont Co., 1967).

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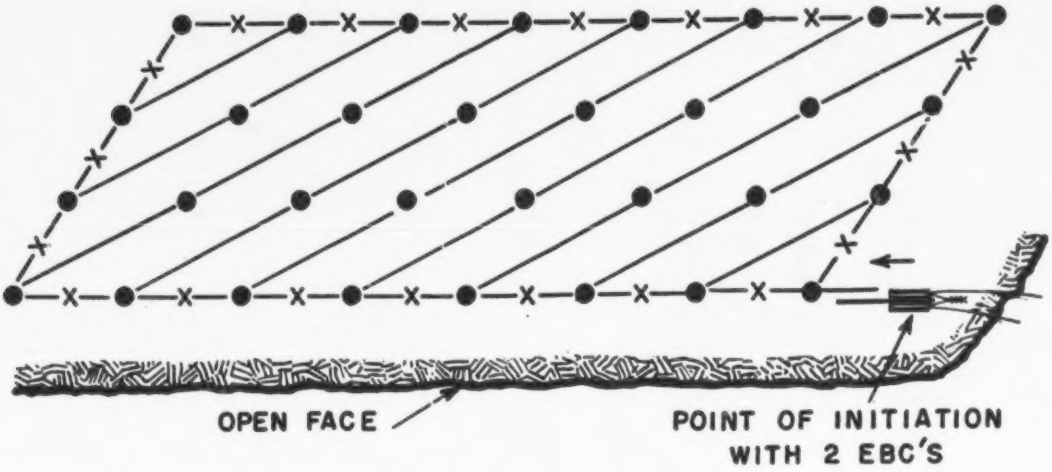
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POINT OF INITIATION WITH 2 EBC'S

Figure 62. Multiple row shot with holes and rows fired in sequence from one end (After DuPont Co., 1967).

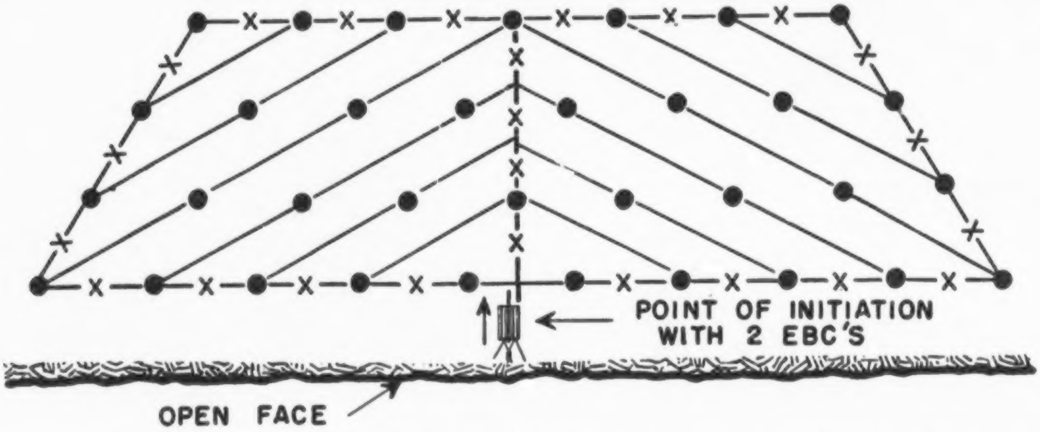
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Figure 63. Multiple row shot in sequence initiated at center front (After DuPont Co., 1967).

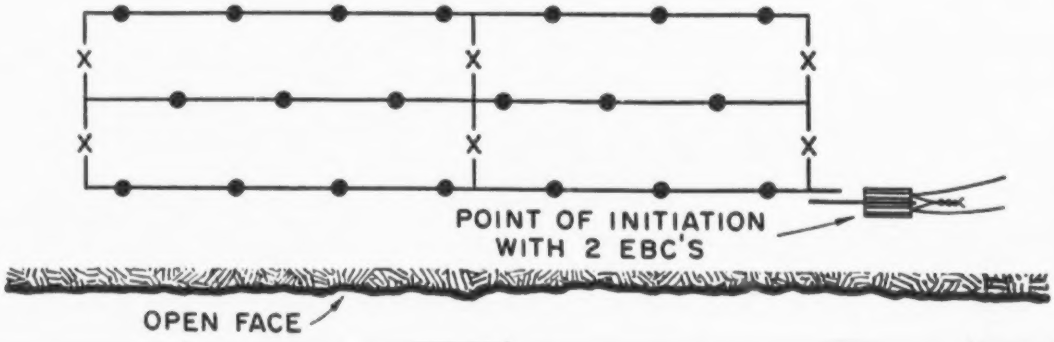
POINT OF INITIATION WITH 2 EBC'S
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Figure 64. Multiple row shot entire row in sequence (After DuPont Co., 1967).

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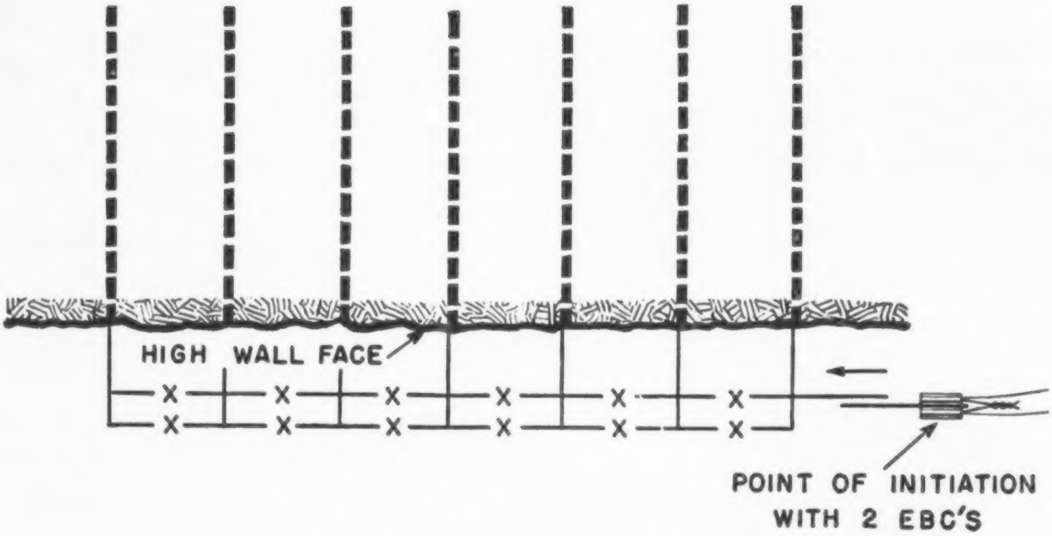
HIGH WALL FACE
X
POINT OF INITIATION WITH 2 EBC'S

Figure 65. Single row horizontal hole shot in bituminous stripping (After DuPont Co., 1967).

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3. When necessary, the primer should be water resistant.
4. A dynamite punch should be used to make the hole for insertion of the blasting caps or detonating cord.
5. Since detonating cord will initiate every cartridge it comes in contact with, care should be taken that it is attached to the first cartridge loaded into the hole.
6. Primer assemblies should be made just before the holes are to be loaded and at the blast area, to reduce the exposure time to hazards such as fire, impact, or electricity.

A booster is an explosive, usually encased in a small metal shell, to improve the performance of another explosive. They are, quite often, placed at intervals along the borehole to reinforce the detonation of the column of blasting agent. Boosters differ from primers in that they do not contain an initiating device. Since boosters are placed between the detonators and blasting agents, they are essential for the ignition of cap insensitive blasting agents.

An explosive is a compound or mixture of compounds which undergoes a rapid, self-propagating exothermic reaction. This reaction is induced by either heat, impact, or shock. When the reaction moves through the explosive at a faster rate than the speed of sound in the unreacted explosive, it is called a detonation. If it is slower, it is called a deflagration. In any event, the products of the reaction, usually gases, exert tremendous pressure during expansion. Since the resultant condition created by the shock front is also dependent upon both detonation velocity and the sonic velocity in the rock, the reaction should be designed so that those velocities are equal.

Figure 66 shows a standard cartridge of explosive in the process of detonating. The right side is undetonated explosive, the left is expanding byproducts of the reaction, and the center represents the area where the explosion, a chemical reaction, is taking place. The explosion is a rapid oxidation reaction in which all elements necessary for the reaction are present in the explosive. The reaction takes place in a very definite zone though, in most cases, the zone is not as wide as is indicated in the figure. The actual width of the zone varies with a coarseness of the explosive particles, from small fractions of a millimeter in the fine-grained high explosives, to several centimeter in bulk, granular blasting agents such as ammonium nitrate/fuel oil (ANFO) mixtures.

Just as rock properties, such as density, porosity, strength, energy absorption, etc., affect fragmentation, there are various explosive properties which also affect the outcome of the fragmentation process. Among these are detonation velocity, density, detonation pressure, borehole presure, and water resistance.

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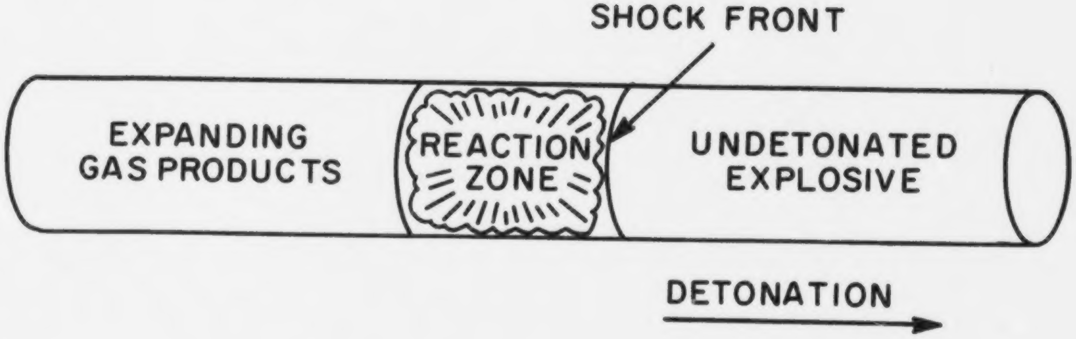
SHOCK FRONT
EXPANDING GAS PRODUCTS
REACTION ZONE
UNDETONATED EXPLOSIVE
DETONATION

Figure 66. A cartridge of explosive undergoing the process of detonation (After Boddorff, 1974).

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Detonation velocity is the speed at which the reaction front moves forward through a cylindrical charge. The velocity ranges between 1676.4 m (5500 ft) and 7620 m (25000 ft) per second. Faster velocities are appropriate in strong rock where a shattering effect is desirable. Slower velocities are applicable where less demand is on rock shattering, whereas a heaving effect is warranted. The following are steps that can be taken in the field to increase detonation velocity (Dick, 1973):

1. Use a larger charge diameter.
2. Increase density.
3. Decrease particle size.
4. Provide good confinement in the borehole.
5. Use a large initiator or primer.

Density refers to the ratio of explosive density to that of water (specific gravity). The ratio for commercial products varies between 0.5 and 1.7. By relating the specific gravity of the explosive to the borehole diameter, the amount of explosive per foot of borehole can be determined. This is particularly useful when matching blasting requirements to the economics of blasting. Figure 67 shows a nomograph to aid in this calculation.

Detonation pressure is the pressure of the detonation wave propagating through the explosive column. It is measured at the rear of the reaction zone (Chapmen-Jouget plane). Detonation pressure can be calculated from the following equation:

$$P = 4.18 \times 10^{-7} DC^2/(1 + 0.80D)$$

where,

P = detonation pressure in kb (1 kb (kilobar) = 100×10^7 pascals)
D = specific gravity
C = detonation velocity in meter per second

Figure 68 also shows the relationship by use of a nomograph. Although the detonation pressure range for commercial explosives varies between 5 and 150 kb, the higher pressures are applicable where a greater shock wave is needed, such as in competent rock.

Borehole pressure is that which is exerted perpendicular to the detonation pressure. Since it is exerted on the borehole walls, it is important in displacing, as well as breaking, the rock. Borehole pressures range from 10 to 60 Kilobar in commercial explosives, where borehole pressures common to ANFO are greater than detonation pressures. In high explosives, however, the converse usually hold true.

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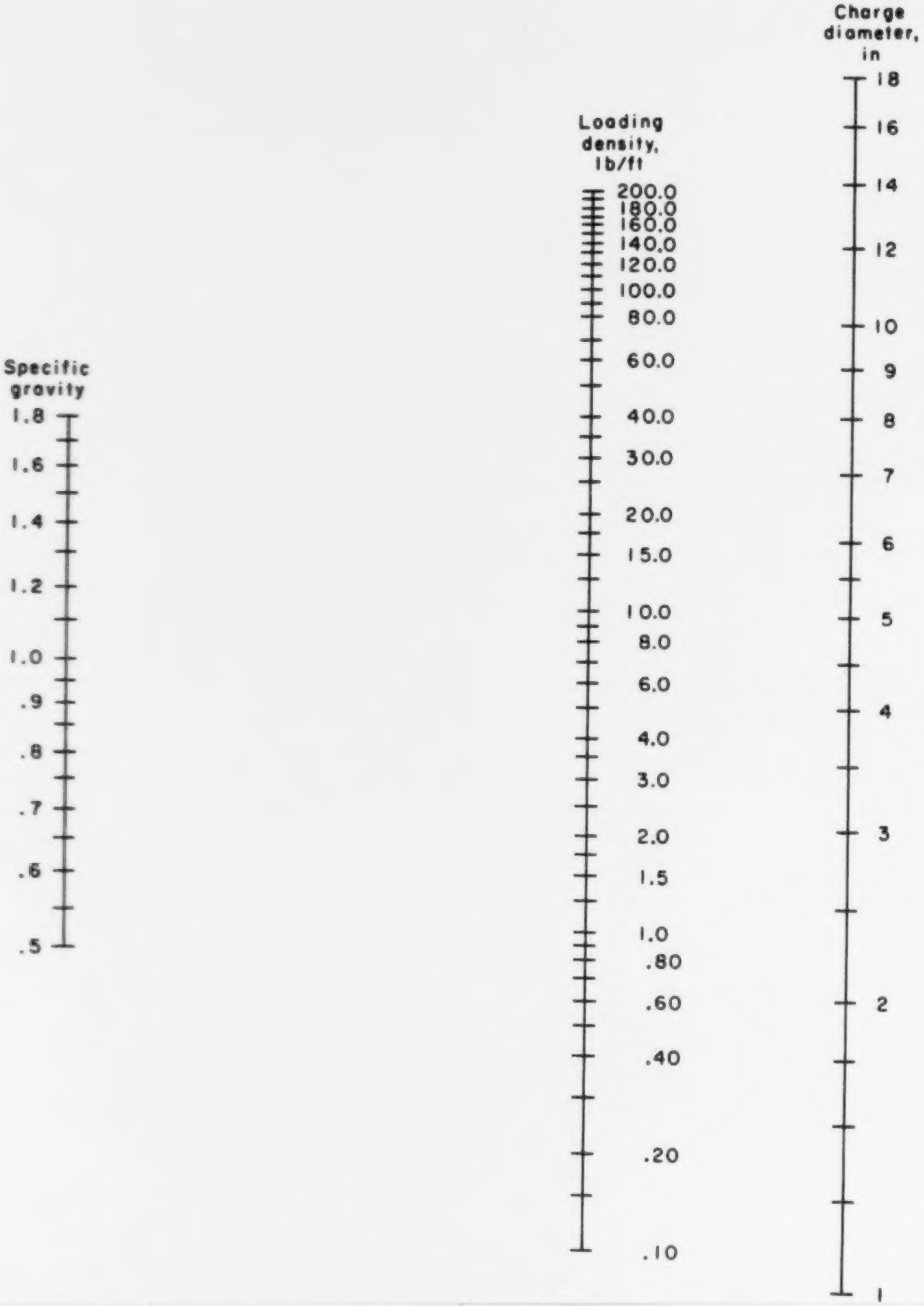

Figure 67. Nomograph for determining loading density (After Dick, 1973).

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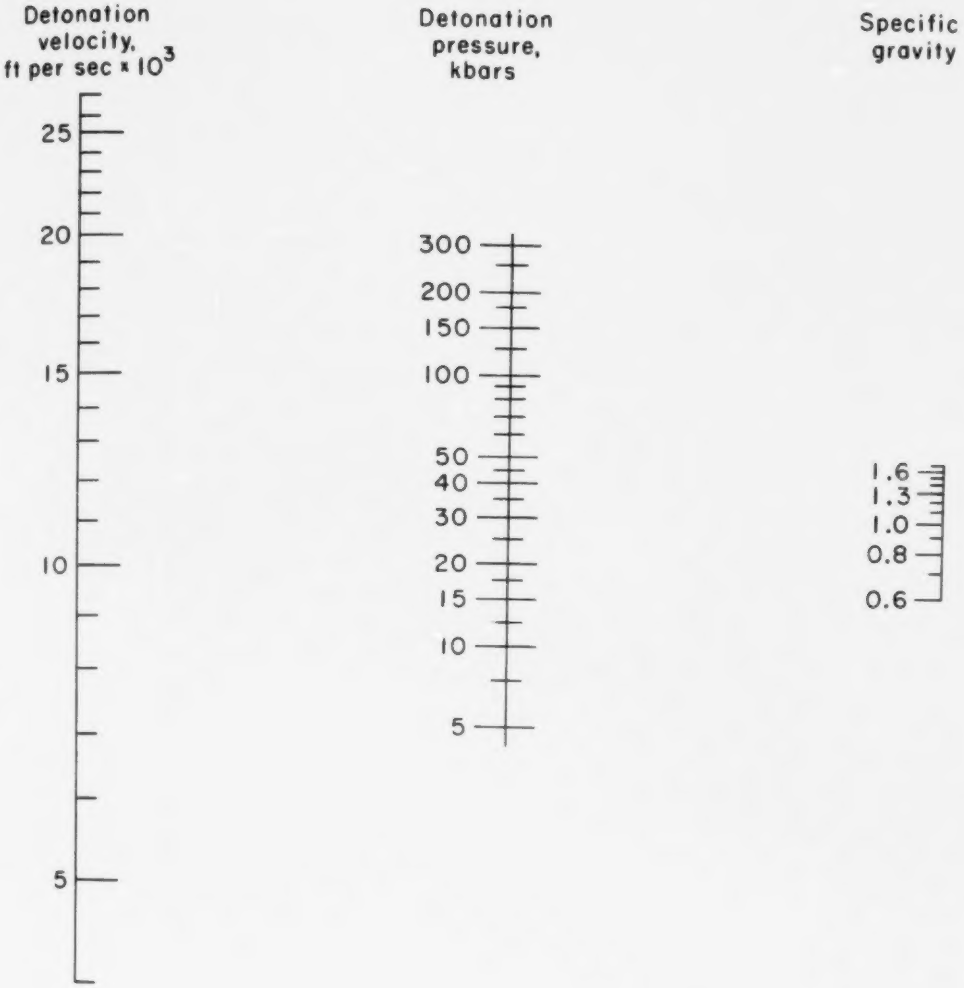

Figure 68. Nomograph for finding detonation pressure (After Dick, 1973).

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Water resistance is the ability of an explosive to withstand exposure to water without losing its sensitivity or efficiency. Gelatin dynamites and slurries have good resistance characteristics. In between these two extremes are the nongelatinized high explosives.

For surface mining applications, there are various types of explosives on the market. However, they are categorized into three groups: 1) high explosives, 2) dry blasting agents, and 3) slurries or water gels. Of all these varieties, dry blasting agents are the most common in coal applications because of cost considerations and safety, and dry blasting agents are cheaper than other forms of explosives and need a primer for detonation. Ammonium nitrate fuel oil, which has become the most common dry blasting agent, accounts for over 75 percent of the nation's consumption. Fifty percent of this figure, 453.6×10^6 Kg (10^9 lbs), is used annually for the surface mining of coal (Condon and Snodgrass, 1974). Due to its overwhelming acceptance, further discussion in this section will be dedicated to dry blasting agents.

A blasting agent is a mixture whose individual components are not classified as explosives and contains a fuel as an oxidizer. Ammonium nitrate fuel oil, though widely used, has only been commercially available since the early 1950's. It was discovered when detonation was observed after mixing ammonium nitrate prills with a certain proportion of a hydrocarbon fuel (typically, 3.78×10^3 cu m (4 qt) of oil to each 45.36 Kg (100 lbs.) ammonium nitrate). This led to the adoption of ANFO as a blasting agent with 60 percent of the effectiveness of dynamite with a cost, presently, of approximately $0.22 per Kg. Even though it represents an added cost, aluminum chips (5 to 30 percent by volume) are sometimes added to the mixture to increase explosive output. Ammonium nitrate fuel oil usually comes to the mine site in one of four manners:

1. As separate ingredients in bulk form for on-site mixing.
2. Premixed in bulk.
3. In paper or polyethylene bags.
4. In cartridges.

Because ANFO is usually mixed on site, there are numerous parameters which must be understood to maximize detonation velocities. Condon and Snodgrass (1974) list them as follows: fuel oil content, confinement, blasthole diameter, density, prill particle sizing, water content, critical diameter, borehole coupling (the percentage of the borehole diameter which is filled with explosives), temperature, and priming system. They examined the impact of primer type and borehole diameter on detonation velocity at two large surface mines in Indiana, since these variables are the easiest to control by mine operators. They concluded:

1. Larger diameter boreholes provide for you more efficient detonation of ANFO and greater energy per pound of powder.
2. The same steady-state velocities were achieved with different types of primers.

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Although the application of ANFO is relatively safe, there are a number of precautions which must be taken into account (Coal Age, 1972):

1. Ammonium nitrate and fuel oil should be mixed in special plants under closely controlled conditions.
2. Locate mixing plants in an isolated area and build them with fireproof materials.
3. Mixed material should be removed from the plant as quickly as possible.
4. No welding or cutting should be done in the plant until cleared of explosive materials.
5. Primers, detonating cord, and caps should be transported in separate vehicles from mixed nitrate and oil. They should be brought together only at the point of use.

Design of Blasting Rounds

Figures 69 and 70 show some of the characteristics and parameters used in the design of blasting rounds. Figure 69 shows those dimensions which are considered for a single hole, and Figure 70 for a total round including:

D_e = Diameter of the explosive in the borehole
B = Burden, or distance from the charge to the nearest free face
S = Spacing, or distance between two holes
H = Length of hole
J = Subdrilling length, or the length the hole is drilled below the floor of the working pit
T = The portion of the hole that does not contain explosive, or collar length
L = Bench height
b = Spacing between rows of holes as measured perpendicular to the free face
s = Separation between adjacent holes in a given row

These abbreviations are used by both Pugliese (1972) and Ash (1970). As both authors point out, b and s are terms used by operators in referring to the drilling grid, and should not be confused with B and S, the burden and spacing which are measured with reference to free faces.

Ash (1963) and Pugliese (1972) concur in the suggestion of five ratios for use in the design of rounds. Assuming that all measurements of length and diameter are expressed in the same units (cm, m, ft, or in.), the ratios relate the dimensions to be designed through the following set of equations:

$$B = K_B D_e \;;\; K_B = \text{Burden ratio} \qquad (1)$$

$$S = K_S B \;;\; K_S = \text{Spacing ratio} \qquad (2)$$

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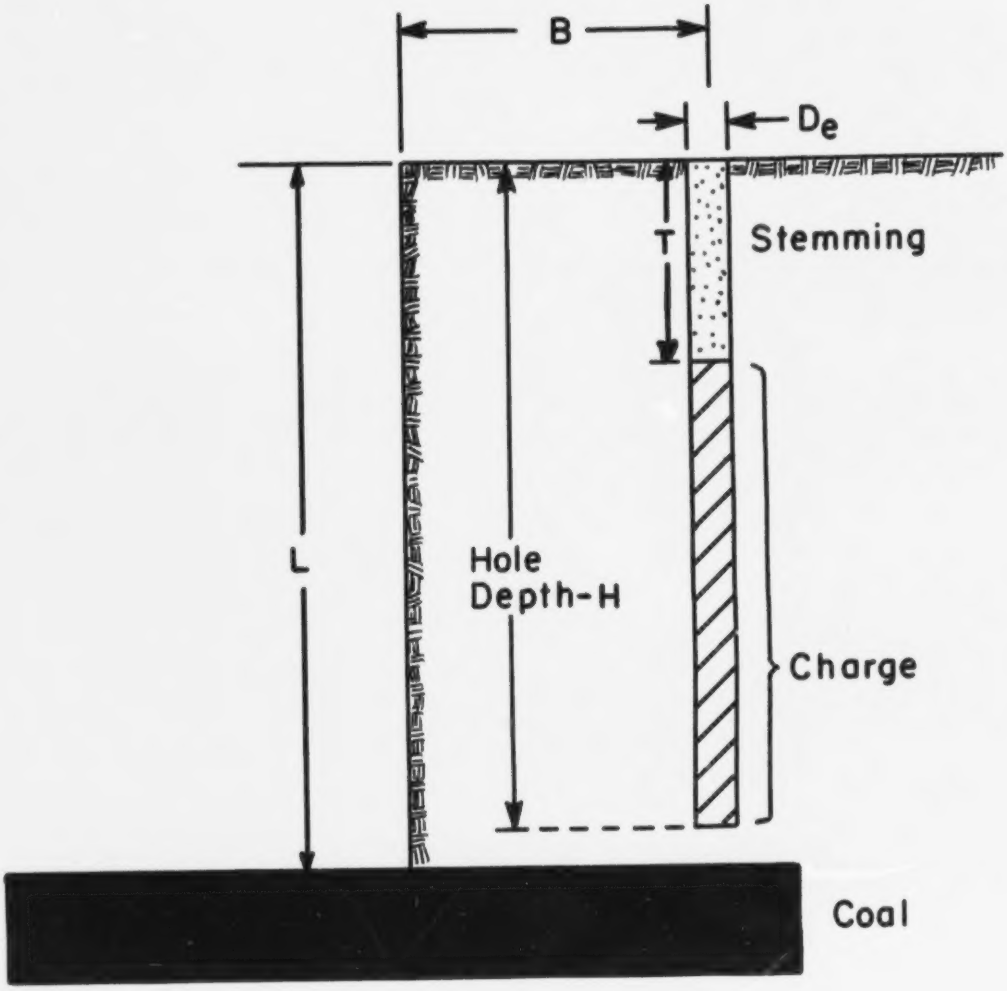
B
D_e
T
Stemming
L
Hole Depth-H
Charge
Coal

Figure 69. Bench cross section view showing D_e, B, H, T, and L (After Pugliese, 1972).

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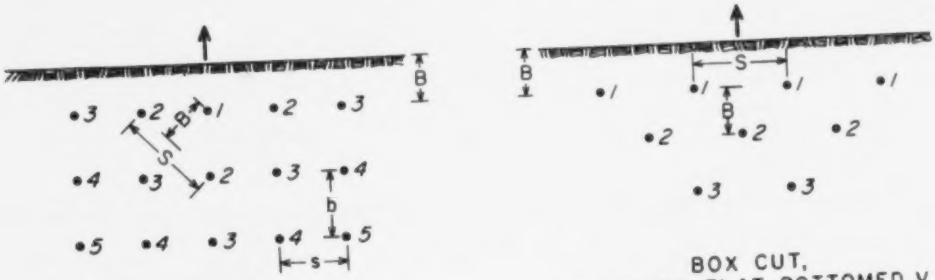
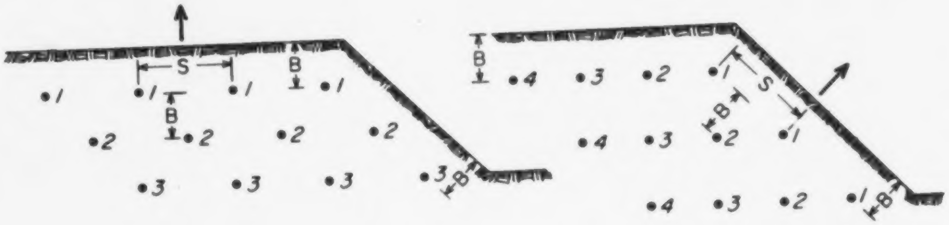


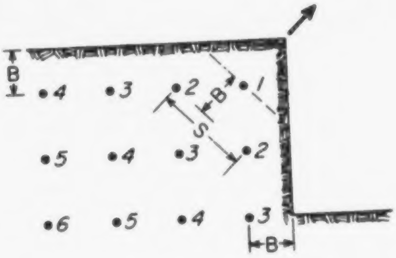
BOX CUT, EXPANDING V
PLAN A

BOX CUT,
EXPANDING FLAT BOTTOMED V
PLAN B

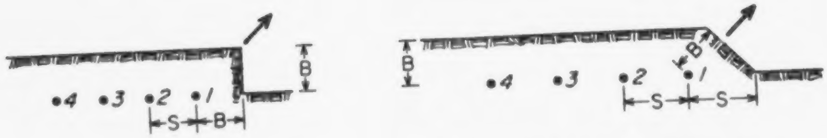


CORNER CUT, ECHELON
PLAN C

CORNER CUT, ECHELON
PLAN D



CORNER CUT, ECHELON
PLAN E



CORNER CUT, ECHELON
SINGLE ROW, PLAN F, S = B

CORNER CUT, ECHELON
SINGLE ROW, PLAN G, S = 1.4B

Figure 70. Generalized blasting patterns showing B, S, b, and s. (Numbers indicate firing sequence) (After Pugliese, 1972).

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$$H = K_H B \quad ; \quad K_H = \text{Hole length ratio} \tag{3}$$

$$J = K_J B \quad ; \quad K_J = \text{Subdrilling ratio} \tag{4}$$

$$T = K_T B \quad ; \quad K_T = \text{Collar distance ratio} \tag{5}$$

From these relationships, all of the critical dimensions of blasting round design can be calculated if appropriate values for the various K ratios are selected. The work of Pugliese and Ash is primarily based on studies in limestone quarrying. While limestone quarrying is conceded as different from overburden removal in surface mining of coal, the relationships still hold and the K values which they have developed and assumed are valid for first assumptions. As is the case with any empirically based design criteria, empirical relationships from literature are usually accepted as a first approximation only, and succeedingly more exact relationships can be developed as actual practice of these design steps is followed.

For calculation of the burden, given a particular D_e, both authors found the following ratios appropriate for limestone and dolomite:

K_B = 30, under average conditions

K_B = 25, for low-density explosives such as ammonium nitrate fuel oil (ANFO)

K_B = 35, for high density slurry explosives and gelatins

The actual K_B ratios are related also to the density of the rock. Limestone and dolomite have a density of approximately 2.7 grams per cubic centimeter. Lower values of K_B should be used for rocks of greater density than 2.7 (e.g., 2.85 and up) or for lower density (e.g., 2.55 and less).

The calculation of spacing is based on K_S values between 1.8 and 2 for simultaneously initiated holes in a given row. Most favorable conditions are with staggered drill patterns and for rows in which all charges are not detonated simultaneously. If exact timing can be achieved, simultaneously fired holes can be designed using K_S values of 3 to 5. These charges must be sufficiently long and exactly timed to allow the proper enhancement of stress effects.

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Spacing is also determined with respect to height and burden relationships, specifically the ratio H/B. Ash (1970) has shown that the relationships

$$S \simeq (B\ H)^{1/2} \qquad \text{for } 2B \leq H < 4B$$

$$S \simeq 2B \qquad \text{for } 4B \leq H$$

will be sufficient. In this situation, the K_S of the 1.8 to 2 is satisfactory, although lower K values can be assigned with H/B less than 3.

If delays are sequenced in a given row, the K_S should range between 1 and 1.2. Square drilling patterns are preferred, and rock will expand in a wedge or in a 45 degree pattern to the original open face (as in Figure 70, Plan A). Considering the ratio of s and b, where b = 1.4B for the regular case, the design relationship

$$S = K_S\ b = K_S\ (1.4B)$$

is appropriate where K_S = 1 to 1.2.

The value of the ratio K_S may be adjusted between 1 and 2 to meet local conditions, particularly the period of delay between charges. Large ratios create spacing so much greater than the burden that the vertical face craters and lumps are left on the pit floor. Small values of K_S, less than 1, create excessive breakage, premature breakage between holes, and progressive loss (rarefaction) from the hole, creating slabs, boulders, and toe problems. The nature of these particular relationships is such that they assume ideal energy balance between charges, so further experimentation with the balance of individual holes may be indicated if such problems are not resolved by varying the K_S ratio.

Hole length calculations are generally based on K_H ratios of 1.5 to 4. Pugliese (1972) cites 2.6 as the typical K_H value. The depth should almost always be greater than or equal to the burden to reduce overbreak and cratering tendencies. Conversely, high values, that is, greater than 4, may create underbreak or bootlegging if the hole is single-primed. Staged priming may allow lengths greater than those indicated by K_H values of 4. No definite limit to K_H can be established without a definite examination of the cratering properties of the specific material being blasted.

The subdrilling length, which is applicable only to quarries and not to coal mines, is generally indicated by K_J values greater than 0.2. Pugliese (1972) suggests 0.3 as preferable to insure a full face and quarry floor. In the case of overburden drilling, however, care is to be taken to limit fracturing of the coal seam and subsequent unnecessary removal of

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coal with the overburden. In some instances, negative K_J values may be applied. Common practice is to drill to the coal and then backfill by 1 to 1.3 m (3 to 4 feet).

The collar distance can be calculated by using a K_T value of 0.7. The uncharged portion of the hole is usually filled with stemming material and is left uncharged to prevent airblast and promote gas confinement within the borehole. In solid rock, Pugliese (1972) suggests that values of K_T less than 1 can lead to cratering, backbreak, and violence, particularly with collar priming.

The various calculations are presented in worksheet form in Table 24. Once the original face and the shape of the cut are selected, the geometrical relationships necessary to work with can be determined from the plan layouts of the cuts. For the given explosive type, blasthole diameter and rock density, the burden is calculated from equation 1. Equation 2 is then solved for the spacing. The hole length can then be checked according to equation 3 and the timing, delay, and sequencing constraints which the operator has established. Finally, the subdrilling length (equation 4), if necessary, can be calculated and the collar distance determined (equation 5).

Blasting Pattern Problems

In the previous section, the design of blasting rounds was viewed from the standpoint of optimum fragmentation of solid rock. However, from a practical standpoint, there are many in situ properties of the overburden and the surrounding environment which also have a great impact on proper design. The major problems which result from these conditions are ineffective fragmentation, vibration, and noise.

Jointed ground complicates the blasting pattern layout. This is due to the fact that the joints act as avenues for the escape of explosive energy. This weakening of the explosive's effectiveness is costly and requires a design to limit the amount of escape. Since blasted material fragments along its structural pattern, it is natural to design the pattern to incorporate this tendency. For example, Figure 71 shows a typical condition in a sedimentary deposit. A rhombic structural pattern is shown with jointing angles at 75 degrees and 105 degrees. If the direction of the blast is designed to be out from the lesser angle (i.e., either N or S), overbreak, coarse fragmentation, and excessive ground vibrations are usually experienced. However, if the blast is designed to be out from the open angle (i.e., W), better fragmentation, a uniform particle size, and a straighter face will result. Therefore, in jointed ground, blasts should be directed out of the open angles, wherever possible.

Ground vibration is an important consideration because of public demands. Damage to surface structures, which is visible evidence of blasting vibration effects, requires constant monitoring of blasting practices. With this in mind, Andrews (1975) developed the following timing guidelines:

1. The minimum delay between holes or corresponding decks in adjacent holes should be 1 millisecond per ft (ms/ft) of burden.

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TABLE 24. WORKSHEET FOR DESIGN OF BLASTING ROUNDS (AFTER PUGLIESE, 1972)

D_e = ____________ in; Cut ____________; Desired rock progression ____________

$B = K_B D_e/12$ = ____________ ft for $K_B \approx$ 30 (average)

$B = K_B D_e/12$ = ____________ ft for $K_B \approx$ ___ (alternative) (Eq. 1)

For staggered pattern, simultaneous timing

$S = K_S B$ = ____________ ft for $K_S \approx$ 2 (average)

$S = K_S B$ = ____________ ft for $K_S \approx$ 1.8 (alternative) (Eq. 2)

For square pattern, sequence timing

$s = K_S\, b = K_S\,(1.4\ B)$= ____________ ft for $K_S \approx$ 1 (average)

$s = K_S\, b = K_S\,(1.4\ B)$= ____________ ft for $K_S \approx$ 1.2 (alternative)

$H = K_H B$ = ____________ ft for $K_H \approx$ 2.6 (average)

$H = K_H B$ = ____________ ft for $K_H \approx$ ___ (alternative) (Eq. 3)

$H_{Min} = K_H B$ = ____________ ft for $K_H \approx$ 1.5 (minimum)

$H_{Max} = K_H B$ = ____________ ft for $K_H \approx$ 4 (maximum)

$J = K_J B$ = ____________ ft for $K_J \approx$ 0.3 (average)

$J = K_J B$ = ____________ ft for $K_J \approx$ ___ (alternative) (Eq. 4)

$T = K_T B$ = ____________ ft for $K_T \approx$ 0.7 (average)

$T = K_T B$ = ____________ ft for $K_T \approx$ ___ (alternative) (Eq. 5)

$L \approx H - J$ = ____________ ft (average or alternative)

$L_{Min} \approx H_{Min} - J$ = ____________ ft (minimum)

$L_{Max} \approx H_{Max} - J$ = ____________ ft (maximum)

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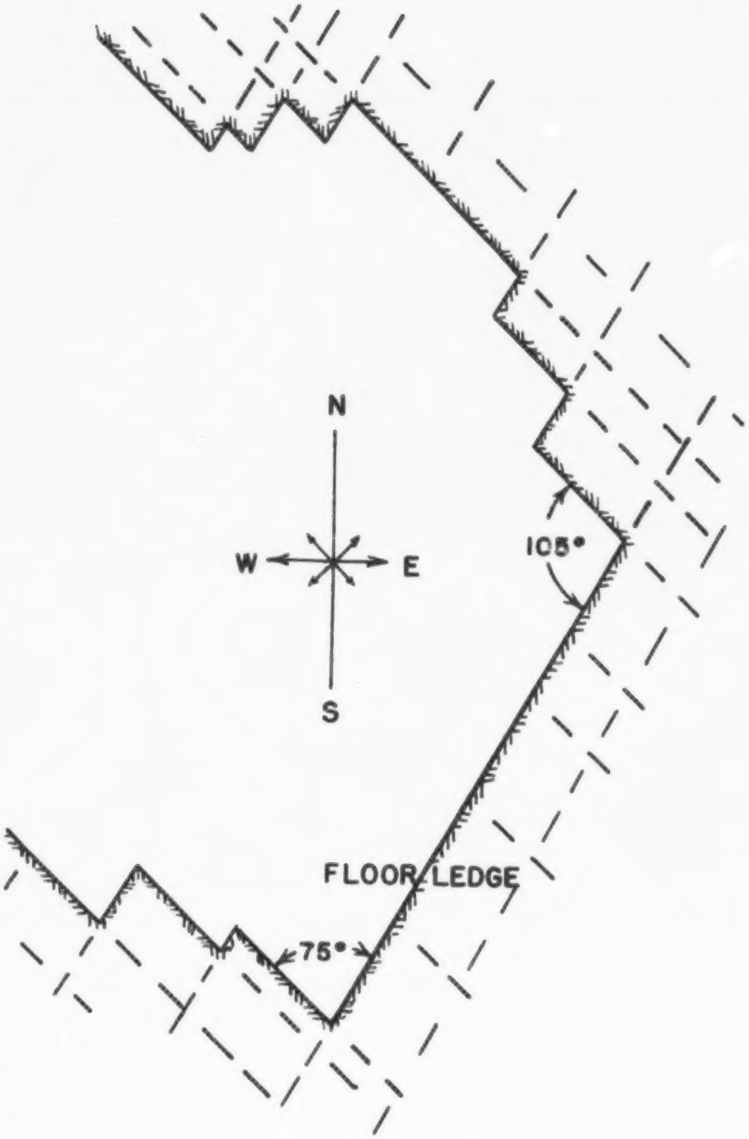
N
W
E
S
105°
FLOOR LEDGE
75°

Figure 71. Representative sedimentary vertical jointing system (After Ash, 1968).

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2. The minimum delay between rows should be greater than the average delay between holes in the same row, to allow sufficient forward and lateral relief for every hole.

Figure 72 portrays the results which led to the first recommendation. In this study (Bergmann, et al, 1974), a low delay ratio produced a large fragment size. However, all the curves tended to flatten out as the delay approached 1 ms/ft of burden. This implies that a greater delay than 1 ms/ft of burden produces little change in fragmentation. The second guideline was established by the comparison shown in Figure 73. If the delay time between rows is equal to the time between holes in the same row (Figure 73), high lateral confinement and poor fragmentation results. In Figure 73, however, a delay time between rows that is twice that between holes in the same row produces a jagged internal free face. This provides both forward and lateral relief for better breakage and reduced ground vibration.

Since any excessive energy from a detonation travels outward through ground, the intensity of the ground vibration is also a function of the distance traveled by the shock wave (i.e., there is a decrease in intensity with an increase in distance). Studies in this area (Nicholls, et al, 1971) reveal that damage to structures by these seismic waves is caused by particle velocity, rather than the displacement or acceleration, of the ground under a structure. Particle velocity is the speed at which the individual unit particles of the ground move as the shock wave is propagated. Figures 74 and 75 show some of the results of the aforementioned studies. Figure 74 shows data for the conclusion reached by the U.S. Bureau of Mines that major damage occurred above a velocity of 19.30 cm/sec (7.6 inch/sec), while minor damage occurred above 13.71 cm/sec (5.4 inch/sec). The Bureau then separated its findings into damage and safe zones, with 5.08 cm/sec (2.0 in/sec) set as the maximum value for the safe zone. Ninety-four percent of the minor damage occurred above this figure, which proved to be statistically acceptable. The value of 5.08 cm/sec (2.0 in/sec) was to be monitored in all three of the orthogonal components of the ground next to the structure. Figure 75 shows, however, that the value is not truly predictive insofar as damage is concerned. It shows that, in some instances, no damage was observed where the particle velocity was greatly in excess of the safe blasting criterion. However, the U.S. Bureau of Mines (USBM) merely feels that damage occurrence increases or decreases as the particle velocity increases or decreases with respect to the value of 5.08 cm (2.0 inch). Therefore, a safe blasting limit of 5.08 cm (2.0 inches) per second for the particle velocity, as measured from any of three mutually perpendicular directions in the ground next to the structure in question, has been used (Grim and Hill, 1974). The maximum peak particle velocity is under reevaluation (see note on Table 25).

For the control of ground vibrations from blasting, a simple formula has been devised relating the blasting charge and proximity to structures (Grim and Hill, 1974). It is given by the equation: $W = (D/50)^2$. In this relationship, W equals the weight of explosives (in pounds) which are detonated at any one instant, and D equals the distance (in feet) from the nearest structure. Instantaneous detonation, in this case, is considered

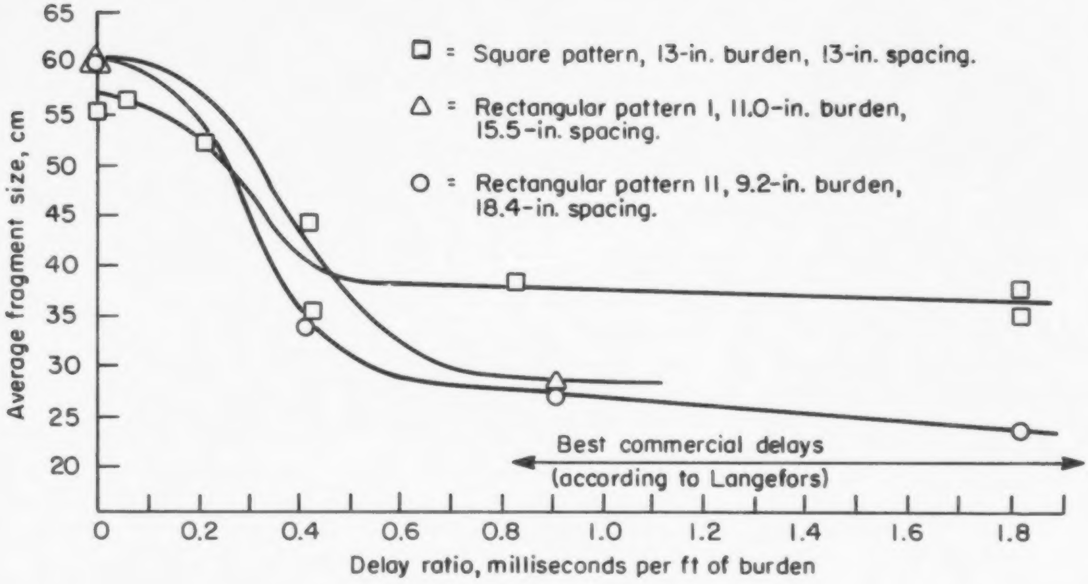

Figure 72. Effect of delay time between shotholes on average fragment size. The same powder factor was used for all shots (After Andrews, 1975).

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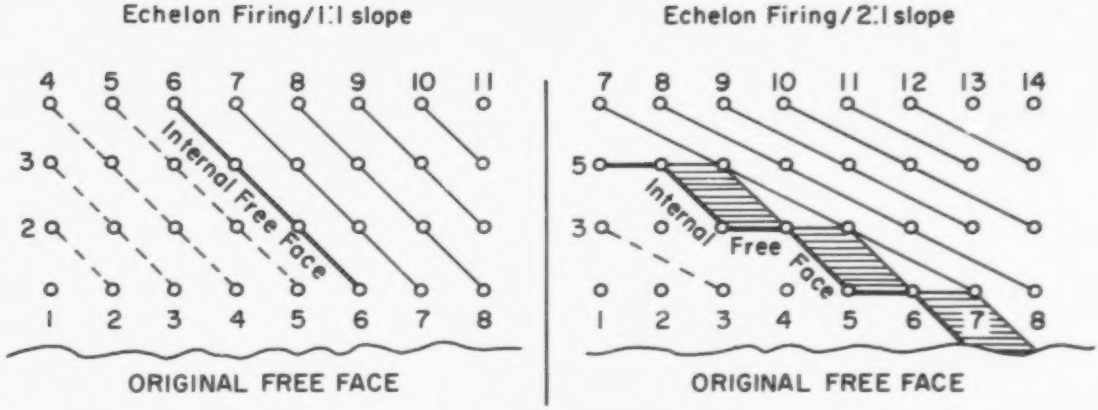
Echelon Firing/1:1 slope
4 5 6 7 8 9 10 11
3
2
1 2 3 4 5 6 7 8
Internal Free Face
ORIGINAL FREE FACE
Echelon Firing/2:1 slope
7 8 9 10 11 12 13 14
5
3
1 2 3 4 5 6 7 8
Internal Free Face
ORIGINAL FREE FACE

Figure 73. Comparison of multiple row patterns using echelon firing on 1:1 and 2:1 slopes (After Andrews, 1975).

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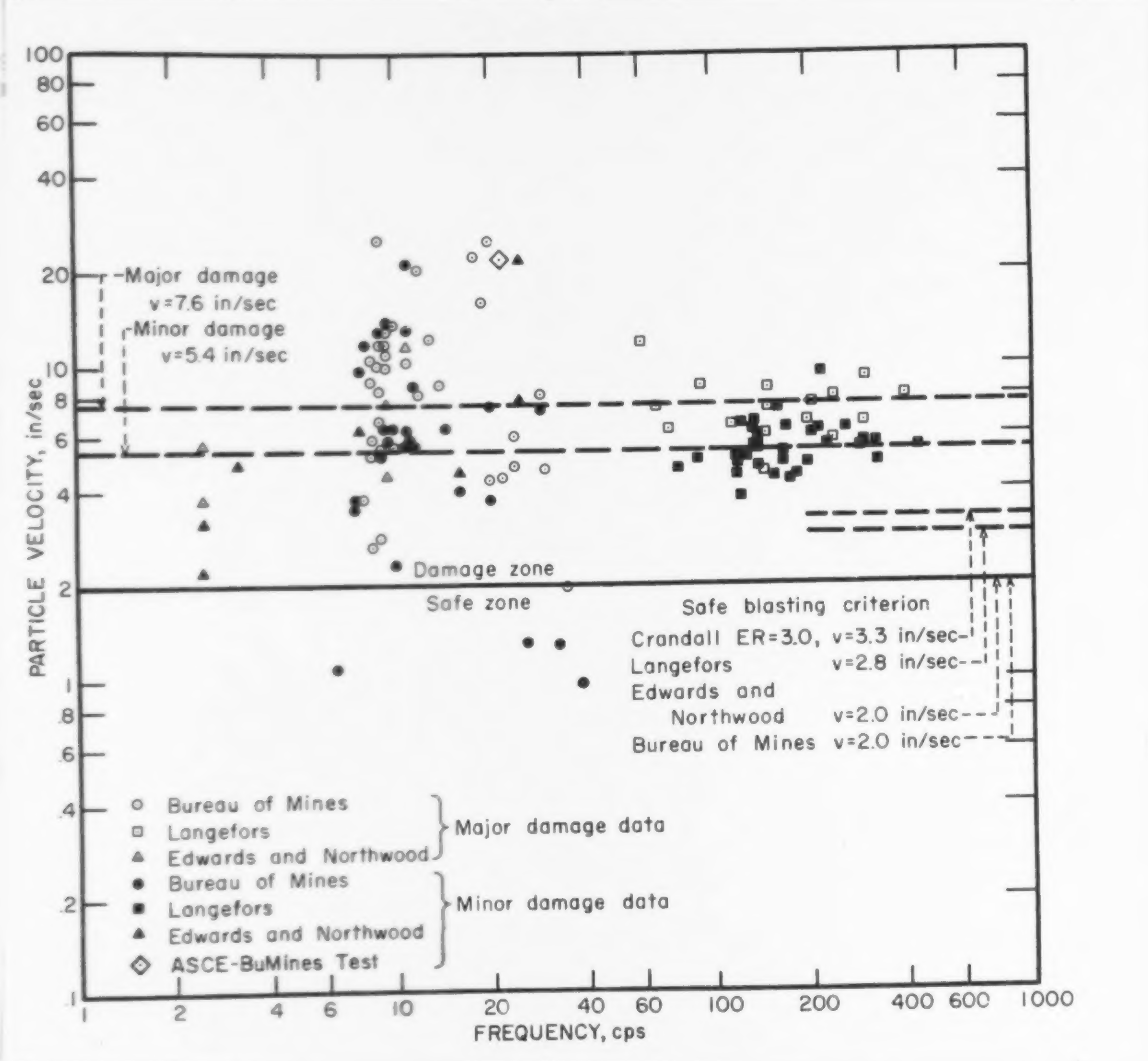

Figure 74. Particle velocity versus frequency with recommended safe blasting criterion (After Nicholls, et al., 1971).

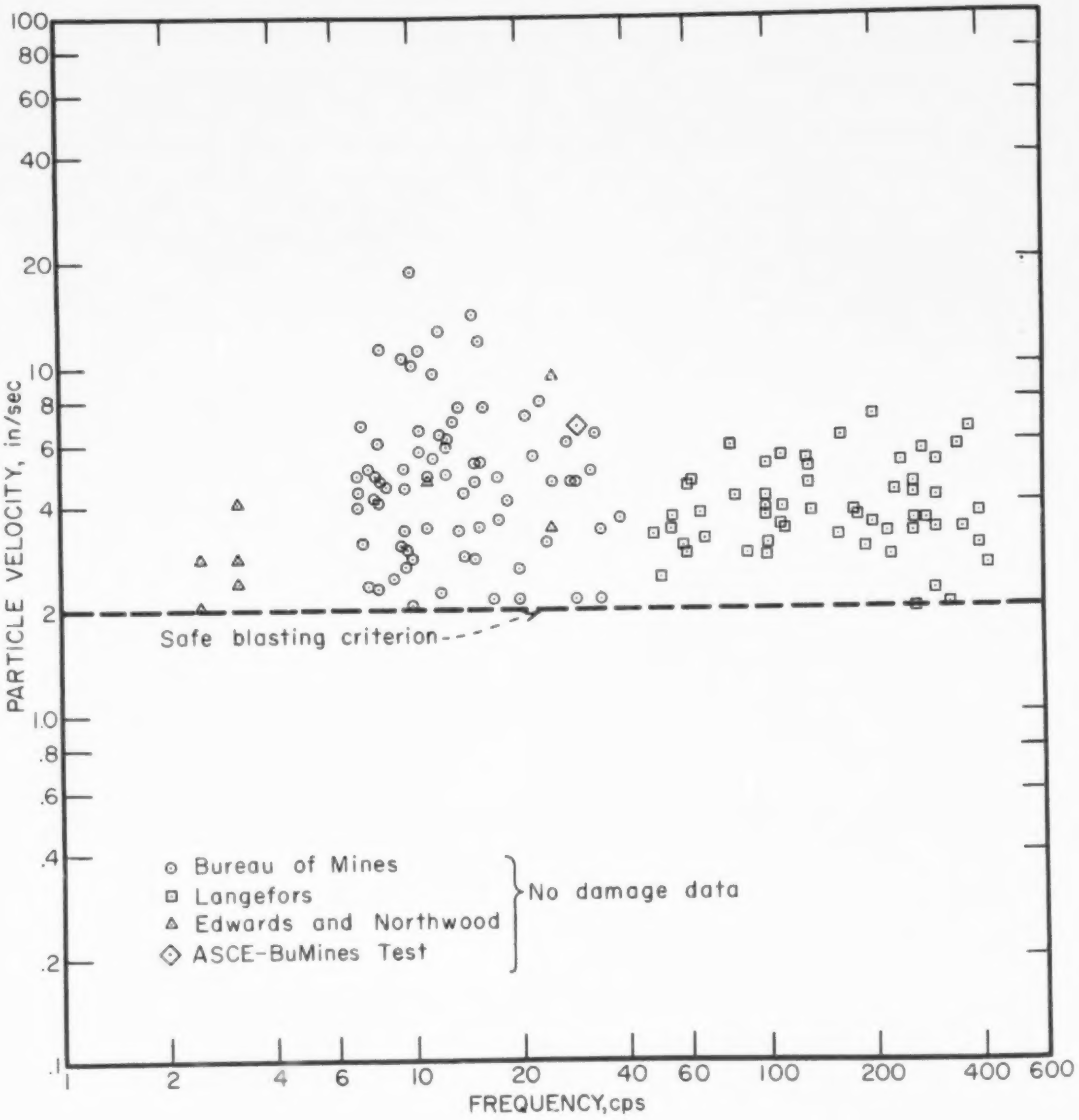

Figure 75. Particle velocity versus frequency for no damage data (After Nicholls, et al., 1971).

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to be all detonation within 8 milliseconds of each other. According to the Federal standards, the formula is revised as $W = (D/60)^2$. The effect of this revision is to reduce the weight of explosive that can be detonated for the same distance, (D). Table 25 and Figure 76 summarize the application of this means of vibration control. Other recommended or practiced blasting patterns are provided in Figures 77 and 78 (McGraw Hill Inc., 1970).

The reduction in noise is beneficial, not because noise creates property damage but because its reduction lessens the social problems faced by operators. Most complaints received by surface mine operators start, initially, because of noise. Although vibrations cause visible physical damage, noise has more of a psychological effect on the nearby population. Since that effect is so difficult to predict, measures must be taken to control the situation as best as possible. As far as noise control is concerned, solutions are generally twofold: 1) attention to blasting techniques and 2) proper evaluation of weather conditions.

Andrews (1975) concluded that, to minimize air blast intensity, the average rate of blast progression along the free face should be less than the velocity of sound in air. If t is the delay time between holes, the ratio d/t, when compared to c, is of extreme importance. When the ratio d/t is less than c, separate shocks radiate in all directions. The resulting air blast behind the face is, therefore, normally weaker and less noisy. When the ratio is equal to or greater than c, a strong air blast is produced due to super-position.

On-site noise control can also be practiced with detonating cord. When cord is used for trunk lines, covering it with 25.40 cm (10 in) of dirt can aid in noise abatement (McGraw Hill Inc., 1970). Further, a new form of low-energy detonating cord has been developed. One-hundred-fifty feet of this cord produces no more noise than 5.08 cm (2 in) of conventional cord (McGraw Hill Inc., 1970).

Weather can also affect noise propagation. Blasting should, when possible, be avoided on foggy, hazy, or smoky days as these conditions are prevalent during temperature inversions. Sound propagation is increased due to these inversions. The following weather conditions, when noted by mine operators, indicate unfavorable conditions for noise control (McGraw Hill Inc., 1970):

1. Relatively high atmospheric pressure that remains static more than 25 hours.
2. Wide daily temperature variations.
3. Poor visibility and light winds early in the morning, followed by cloudiness.
4. Clear, somewhat hazy days with little wind and fairly constant temperatures.

TABLE 25. MAXIMUM EXPLOSIVE CHARGES USING SCALED DISTANCE FORMULA

Distance to nearest residence building or other structure (ft)	Maximum explosive charge (to be detonated) (lb)
300	25
350	34
400	44
500	69
600	100
700	136
800	178
900	225
1,000	278
1,100	336
1,200	400
1,300	469
1,400	544
1,500	625
1,600	711
1,700	803
1,800	900
1,900	1,002
2,000	1,111
2,500	1,736
3,000	2,500
3,500	3,403
4,000	4,444
4,500	5,625
5,000	6,944

Note: In all blasting operations, except as otherwise authorized, the maximum peak particle velocity shall not exceed 1 inch per second at the location of any dwelling, public building, school, church, or commercial or institutional building. Peak particle velocities shall be recorded in 3 mutually perpendicular directions. The maximum peak particle velocity shall be the largest of any of the three measurements. The maximum peak particle velocity allowed may be reduced if it is determined that a lower standard is required because of density of population or land use, age or type of structure, geology or hydrology of the area, frequency of blasts, or other factors.

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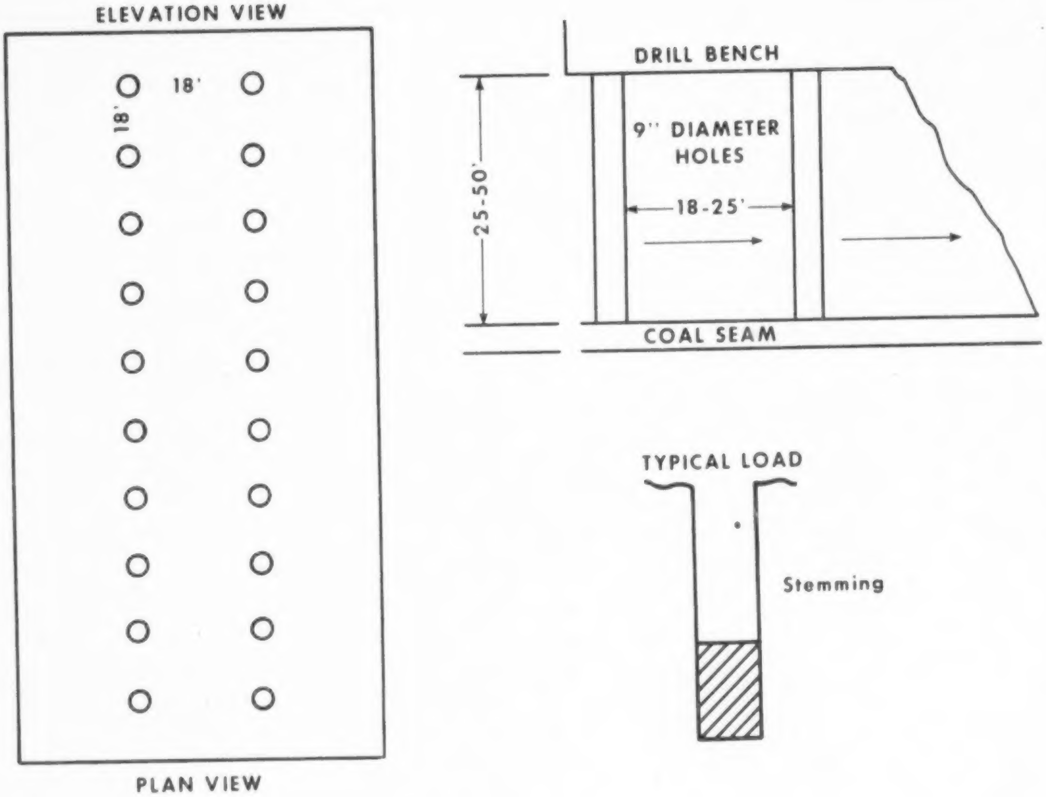
ELEVATION VIEW
18'
18'
PLAN VIEW
DRILL BENCH
9" DIAMETER HOLES
18-25'
25-50'
COAL SEAM
TYPICAL LOAD
Stemming

Distance to nearest dwelling = approximately 5000 feet.
Amount of explosive per hole = 340 lb.

Figure 76. Proposed blasting plan (example) (After Grim and Hill, 1974).

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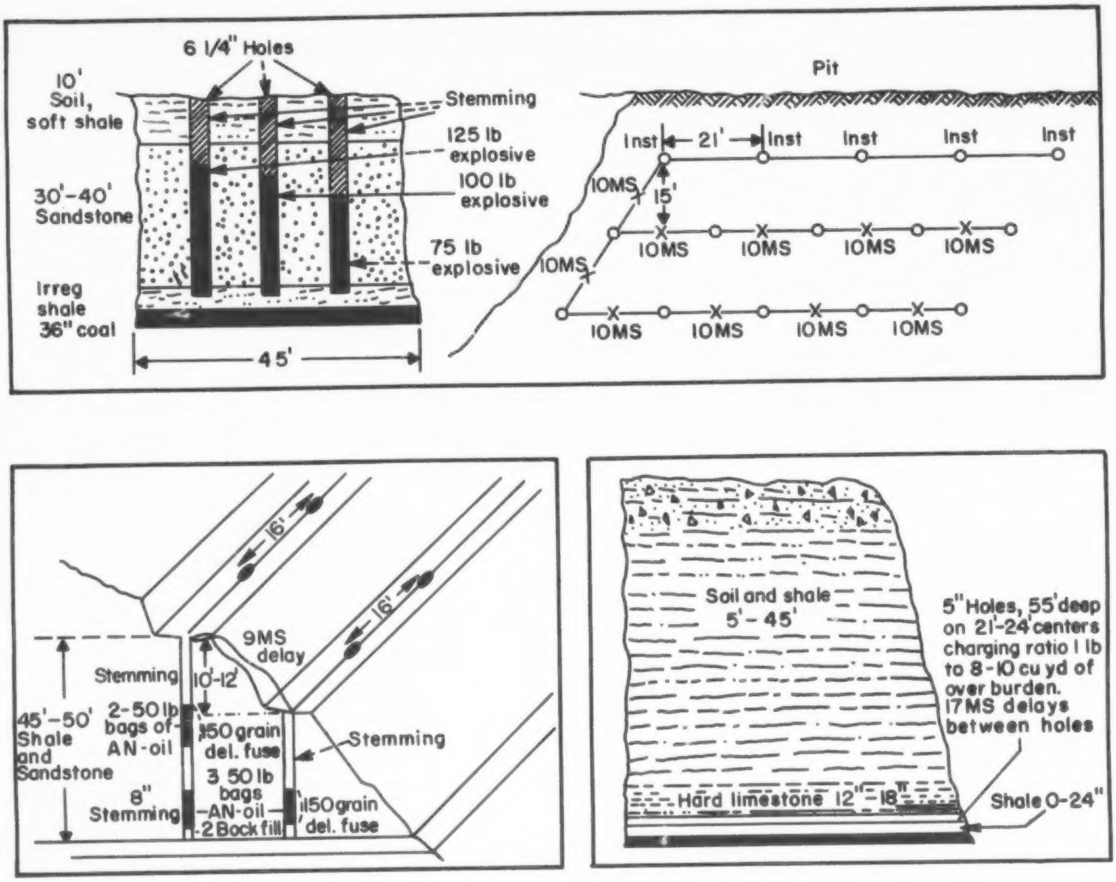
6 1/4" Holes
10' Soil, soft shale
Stemming
125 lb explosive
100 lb explosive
30'-40' Sandstone
75 lb explosive
Irreg shale
36" coal
45'
Pit
Inst
21'
Inst
Inst
Inst
Inst
15'
IOMS
IOMS
IOMS
IOMS
IOMS
IOMS
IOMS
IOMS
IOMS
IOMS
IOMS
16'
16'
9MS delay
Stemming
10'-12'
45'-50' Shale and Sandstone
2-50 lb bags of AN-oil
150 grain del. fuse
Stemming
3 50 lb bags AN-oil
8" Stemming
2 Bock fill
150 grain del. fuse
Soil and shale 5'-45'
5" Holes, 55' deep on 21'-24' centers charging ratio 1 lb to 8-10 cu yd of over burden. 17MS delays between holes
Hard limestone 12"-18"
Shale 0-24"

Figure 77. Some commonly used drilling and blasting patterns (McGraw-Hill, Inc., 1970).

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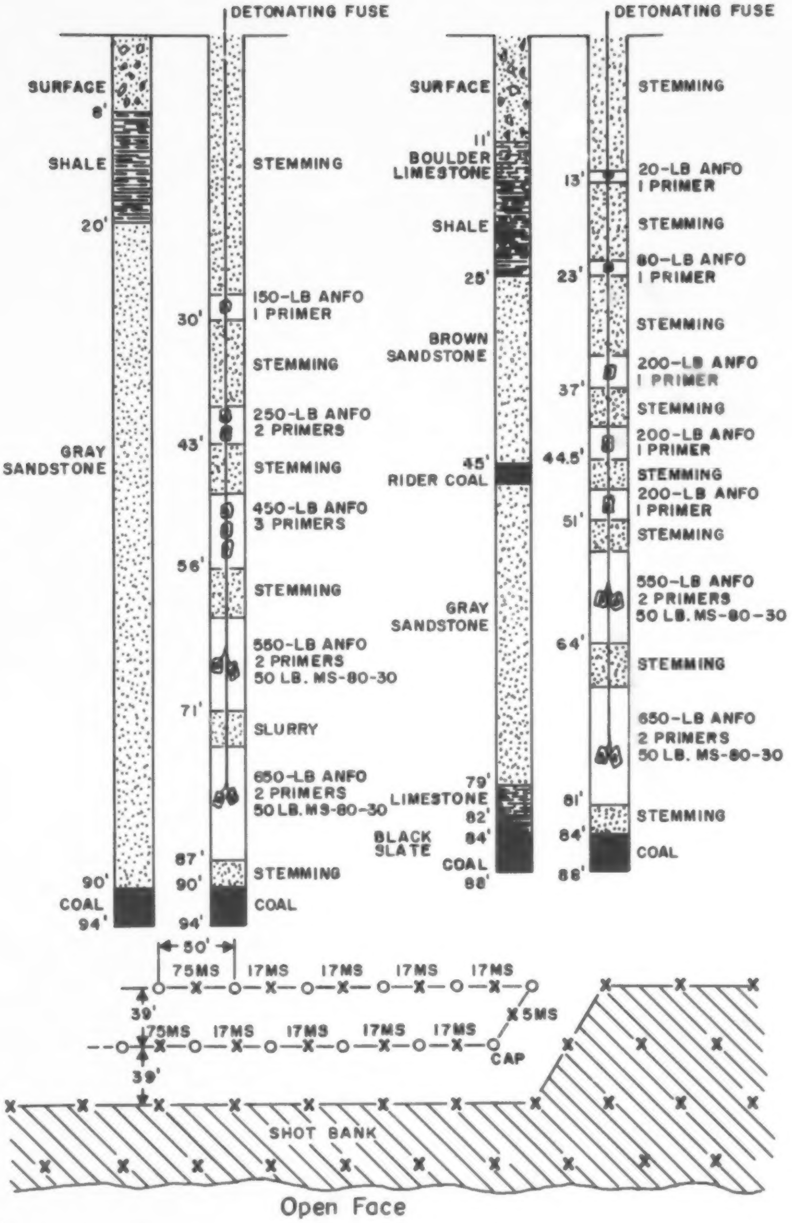
DETONATING FUSE
SURFACE
8'
SHALE
20'
STEMMING
150-LB ANFO
I PRIMER
30'
STEMMING
250-LB ANFO
2 PRIMERS
43'
GRAY SANDSTONE
STEMMING
450-LB ANFO
3 PRIMERS
56'
STEMMING
550-LB ANFO
2 PRIMERS
50 LB. MS-80-30
71'
SLURRY
650-LB ANFO
2 PRIMERS
50 LB. MS-80-30
87'
90'
STEMMING
90'
COAL
94'
94'
COAL
DETONATING FUSE
SURFACE
STEMMING
11'
BOULDER LIMESTONE
13'
20-LB ANFO
I PRIMER
SHALE
STEMMING
28'
23'
80-LB ANFO
I PRIMER
STEMMING
BROWN SANDSTONE
200-LB ANFO
I PRIMER
37'
STEMMING
200-LB ANFO
I PRIMER
45'
44.5'
RIDER COAL
STEMMING
200-LB ANFO
I PRIMER
51'
STEMMING
550-LB ANFO
2 PRIMERS
50 LB. MS-80-30
GRAY SANDSTONE
64'
STEMMING
650-LB ANFO
2 PRIMERS
50 LB. MS-80-30
79'
LIMESTONE
82'
81'
STEMMING
BLACK SLATE
84'
84'
COAL
COAL
88'
86'
50'
75MS
17MS
17MS
17MS
17MS
39'
5MS
75MS
17MS
17MS
17MS
17MS
CAP
39'
SHOT BANK
Open Face

Figure 78. Metalized slurry is used to improve blasting results with ANFO (McGraw-Hill, Inc., 1970).

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SUMMARY

This chapter has briefly reviewed the fragmentation practices in surface coal mining. Vertical rotary blast hole drills are the most common for overburden drilling. Ammonium Nitrate Fuel Oil is the explosive that is widely used. It is common in area mining for the drilling and blasting unit operation to be in advance of stripping by one or more cuts. In contour mining, where small loaders load onto trucks, the size of the blasted overburden becomes important. Therefore, the powder factor in contour operations is very low (varying from 1.68 cu m per Kg (1 cu yd per lb) of ANFO to 4.21 cu m per Kg (2.5 cu yd per lb) of ANFO). Important environmental considerations on which blasting designs must be based are noise, damage to surface structures, and protection from fly rocks.

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SECTION 6

SURFACE MINE HAULROAD DESIGN

INTRODUCTION

For the purposes of this report, coal mine haulroads are defined as roads constructed and/or used by a mine operator, which end at the pit or the bench. These roads may cover large areas of ground, and, according to a U.S. Department of the Interior report (1967), constitute approximately 10 percent of the total area directly disturbed by the surface mining operation.

According to the Surface Coal Mining and Reclamation Operations, Permanent Regulatory Program (Public Law 95-87, Title 30, Chapter VII, Subchapter A), road means a surface right-of-way for purposes of travel by land vehicles used in coal exploration or surface coal mining and reclamation operations. A road consists of the entire area within the right-of-way, including the roadbed, shoulders, parking and side area, approaches, structures, ditches, surface, and such contiguous appendages as are necessary for the total structure. The term includes access and haulroads constructed, used, reconstructed, improved, or maintained for use in coal exploration or surface coal mining and reclamation operations, including use by coal-hauling vehicles leading to transfer, processing, or storage areas. The term does not include pioneer or construction roadways used for part of the road construction procedure and promptly replaced by a Class I, Class II, or Class III road located in the identical right-of-way as the pioneer or construction roadway. The term also excludes any roadway within the immediate mining pit area.

(a) Class I Road means a road that is utilized for transportation of coal.

(b) Class II Road means any road, other than a Class I Road, planned to be used over a 6-month period or longer.

(c) Class III Road means any road, other than a Class I Road, planned to be used over a period of less than 6 months.

Subchapter K of the above regulatory program, specifically sections 816.150 to 816.175, details the general considerations, location, design and construction, drainage, surfacing, maintenance and restoration of the Class I, II and III roads. Movement of coal, equipment and personnel within the mine plan area may require, other than roads, railroad loops, spurs, sidings, surface conveyor systems, and other transportation facilities. General standards are set forth in section 816.810 to ensure the minimization of the adverse effects to hydrology, fish and wildlife and their habitats, and public and private property as a result of the design, construction and utilization of these transportation facilities. It is not the purpose here to detail all the provisions of the regulations but a brief summary of the important requirements for Class I roads will be provided.

CLASS I ROADS

General

Class I roads should be located, insofar as possible, on ridges or on the most stable available slopes to minimize erosion. No part of this road may be located in the channel of an intermittent or perennial stream, and excepting for approved stream fords, all stream crossings shall be made using bridges, culverts or other structures which are designed, constructed and maintained as per the provisions of the law. All Class I roads shall be removed and the land affected regraded and revegetated unless the retention of the road is approved as part of the post mining land use or as being necessary to control erosion.

Design and Construction

The design and construction of Class I roads shall be approved by a registered qualified professional engineer and shall incorporate the demand for mobility and travel efficiency, based on geometric criteria (vertical alignment, horizontal alignment, road cuts, and road embankments) appropriate for the volume of traffic and the weight and speed of vehicles that will be used for coal haulage. The overall grade shall not exceed 10 percent, and the maximum pitch grade shall not exceed 15 percent. Considerable importance has been attached to ensuring that adequate drainage facilities are included in the design and construction of these roads. Specifically, ditches, culverts, and bridges must be provided so as to minimize 1) the erosion from the roadway, 2) the siltation of adjacent streams, 3) the adverse impact on fish and wildlife habitat and 4) the instability of the roadway to protect the welfare of the public. In general, the water control system shall be designed to safely pass the peak run-off from a 10-year, 24-hour precipitation event or a greater event if required by the regulatory authority.

Maintenance

Class I roads should be maintained so as to meet the approved design standards at all times. Maintenance will include repairs to the road surface, minor reconstruction of road segments as necessary, brush removal and revegetation. Roads that have been damaged by floods or earthquakes will not be used until the damaged areas have been reconstructed.

OTHER TRANSPORTATION FACILITIES

These facilities shall be designed, constructed, and maintained, and the area restored to 1) prevent, to the extent possible, damage to fish, wildlife, and other environmental values, 2) prevent contribution of suspended solids into streamflow or runoff outside the permit area, 3) control water quality and quantity degradation, 4) minimize erosion, siltation, and air pollution, and 5) prevent damage to public and private property.

This chapter will briefly describe the steps necessary for the design of haulroads, since a well-designed haulroad will minimize the problems enunciated before.

SURFACE MINE HAULROAD

The distinguishing feature that sets a mine haulroad apart from any other road is the duration of its usefulness. In general, most roads are designed with a specific lifetime in mind or with perpetual existence assumed. Haulroads may be in use for a period of days to many years, depending upon the mine plan and other related factors. Therefore, it is important to know the estimated time a proposed haulroad will be in use. The use of very expensive surfacing material on a short-duration haulroad would not be justified if the road could be maintained with an additional road grader for less total cost. In fact, in most surface coal mines, haulroads are maintained in mined-out pits and through spoil banks for considerable distances before the permanent section of the haulroad is reached. For this reason, the total mine plan should be considered when planning haulroads.

HAULROAD DESIGN

The design principles used for haulroads differ little from that for any other road; however, the roads for many large surface mines today have to be designed to carry loads far in excess of the loads found on such highways as the interstate system. With the introduction of larger equipment, such as off-highway trucks, there presently seems to be no weight limit with trucks weighing 355.6×10^3 Kg (350 tons) now in existence. Unlike the interstate highway system which limits the loads on its roads, haulroads may be subjected to larger loads simply because the mine operator is forced to purchase larger equipment in order to remain competitive. The ability to increase the bearing capacity of hualroads with minimum cost is a design feature that warrants some thought during the initial design stages.

Often, the road-building material is a mine product, crushed overburden and/or bottom ash. Therefore, the construction of subbase should be on the basis of the results of the soil survey and the design of traffic loads. In area mining, since production from a mining site lasts for several years, the topography is generally gentle, the hauling equipment used is usually large compared to that used in contour mines, and the roads are usually better designed. In contour mines, the roads are long and usually have to be laid on severe grades. Water seepage from the hillsides adds to the maintenance problem. It is also not uncommon for haulroads to be maintained through spoil piles and old pits. Even without these, the fugitive dust problem with vehicular traffic demands that the roads be wetted and maintained in good condition to keep ahead of grading and surfacing problems.

Regular checking of ditches and culverts for drainage and incorporation of $CaCl_2$ or $NaCl_2$ into the road surface will add to the mechanical stability of the road surface. Adequate facilities for watering and water trucks must be available in the mines.

When a haulroad is abandoned, steps must be taken to minimize erosion and establish a vegetative cover. For complete abandonment, culverts and other structures are removed, and the natural drainage pattern is restored. Side ditches should be obliterated, and properly spaced grade dips or water bars should be constructed to handle roadway cross drainage. A water bar must be placed at the head of all pitch grades, regardless of spacing. All road surfaces must be ripped, treated with soil amendments, seeded with grasses, legumes, and trees, and mulched. Seeding will help stabilize the abandoned roads, provide food for wildlife, and improve the aesthetics.

ALIGNMENT

In general terms, alignment refers to the design of the geometric elements of the haulroad. Such things as horizontal and vertical curves, site and stopping distance, superelevation, and maximum grade all come under this general term. Any standard text on highway construction can be referred to for more information (Meyer, 1969). Specifically, with regard to surface mines, a recent study by Skelly and Loy (1976) and the U.S. Bureau of Mines Information Circular 8758 (1977) are useful references. Murray (1978) provides a computer program to calculate the best alignment of a road (i.e., road grade) for a given topographical profile, taking into consideration the truck capacity and speed-rimpull characteristics.

Superelevation refers to the sloping of the road toward the center of the curve. Superelevation must be provided on curves to reduce the side friction between the tires and the road. It is recommended that the superelevation should never be less than 0.20 meter per meter because most road surfaces are crowned that much to provide drainage of the road from side to side. Because of the danger of sliding off the inside of a steeply superelevated highway when traveling slower than the design speed, the superelevations in Table 26 are recommended.

Horizontal curves should be well laid out. When horizontal curves are excessively sharp, the road surface should be widened to account for the added width of passing vehicles (Figure 79).

It would be impractical to build a haulroad with an abrupt change in grade. When negotiating changing grades (i.e., vertical curves), one must consider two other aspects of vertical design, namely sight distance and stopping distance. It is recommended that a vertical curve should never be less than 30.48 meter (100 feet) in length.

By keeping haulroads at their minimum slopes, cycle times will be increased and wear and tear on the equipment will be reduced. Although

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TABLE 26. RELATIONSHIP BETWEEN SPEED AND RADIUS OF A CURVE AND THE SUPERELEVATION RECOMMENDED

	SPEED/MPH							
RADIUS/FT	10	15	20	25	30	35	40	50
50	0.04	0.04						
100	0.04	0.04	0.04					
150	0.04	0.04	0.04	0.05				
250	0.04	0.04	0.04	0.04	0.06			
300	0.04	0.04	0.04	0.04	0.05	0.06		
600	0.04	0.04	0.04	0.04	0.04	0.05	0.06	
1000	0.04	0.04	0.04	0.04	0.04	0.04	0.05	0.05

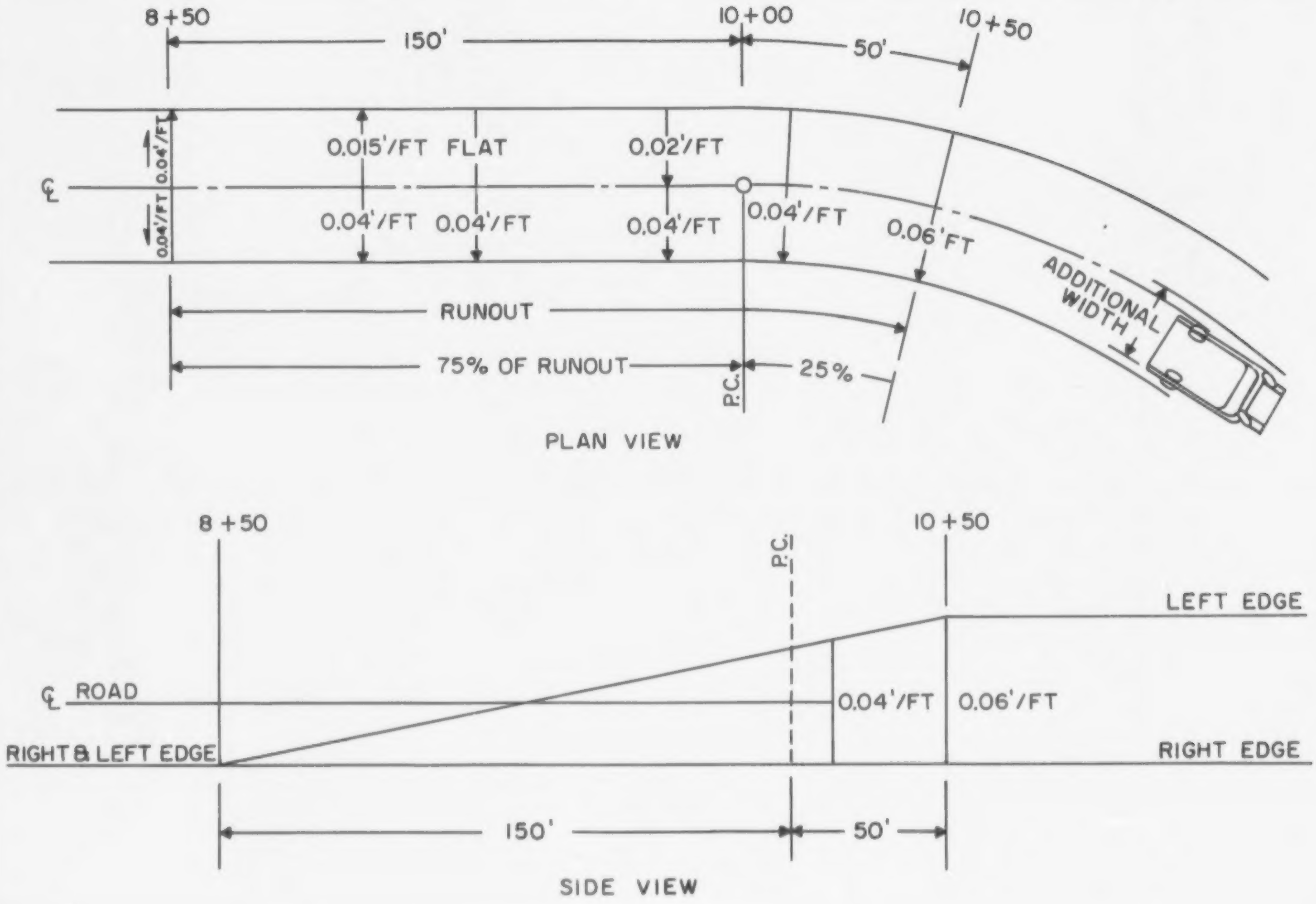
8 +50
150'
10 +00
50'
10 +50
0.015'/FT
FLAT
0.02'/FT
0.04'/FT
0.04'/FT
0.04'/FT
0.04'/FT
0.04'/FT
0.04'/FT
0.06'FT
ADDITIONAL WIDTH
℄
RUNOUT
75% OF RUNOUT
25%
P.C.
PLAN VIEW
8 +50
10 +50
P.C.
LEFT EDGE
℄ ROAD
0.04'/FT
0.06'/FT
RIGHT & LEFT EDGE
RIGHT EDGE
150'
50'
SIDE VIEW

Figure 79. Superelevation runout considerations.

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it would be impractical to design haulroads so all vehicles could operate at their maximum speed at all times, some compromise between percent grade and cycle time can be reached.

The stopping distance (SD) for various size vehicles can be computed using the following formula:

$$SD = [1/2\ gt^2 \cdot \mathrm{Sin}\ \Theta + V_o t] + [\frac{(gt\ \mathrm{Sin}\ \Theta + V_o)^2}{2g(U\ MIN - SIN\ \Theta)}$$

where

SD = Stopping distance in meter.
g = Acceleration of gravity, 9.81 m/sec^2 (32.2 ft/sec^2)
t = Reaction time of the driver
Θ = Angle of descent in degrees
V_o = Initial speed before breaking
$U\ MIN$ = Coefficient of friction between break shoe and drum

Sight distance is the extent of peripheral area visible to the vehicle operator. As stated earlier, sight distance should never be less than stopping distance. The easiest and possibly the most used method of determining sight distance is to graphically construct the vertical curve, and then from the known stopping distance, see if the sight distance is adequate.

ROADWAY BEARING CAPACITY

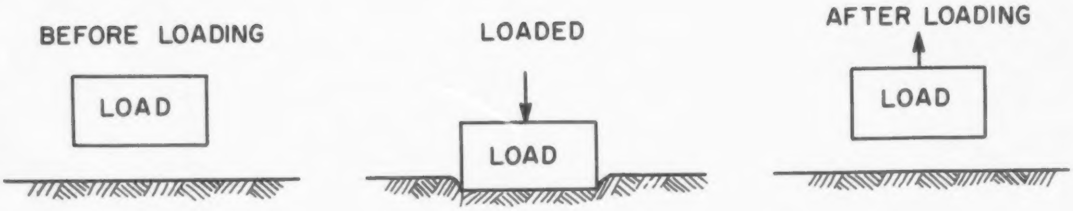
Once the geometric parameters of the haulroad have been established and the route of the road known, it will be necessary to determine the load-bearing capacity of the various materials used in constructing the road.

Most soil materials are subject to three types of deformation, namely elastic, consolidation, and plastic (Figure 80). The elastic deformation will rebound to its original shape when the load is removed. The consolidation deformation will not rebound, but leave the loaded area denser than the unloaded area. Plastic deformation occurs when the water and air in the pores of the soil combine with loading forces to displace the roadway material (see Figure 80).

PLASTIC DEFORMATION

Plastic deformation is the most damaging type of deformation to the haulroad surface, and is the deformation that the road must be designed to withstand.

The initial step in this design is to determine the loads that the road is expected to withstand. This is done by computing the wheel loads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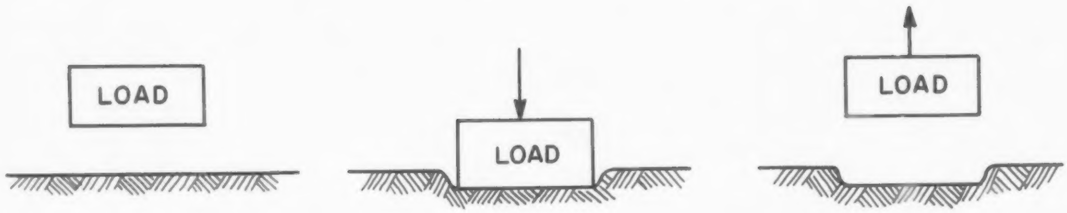
ELASTIC DEFORMATION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CONSOLIDATION DEFORMATION

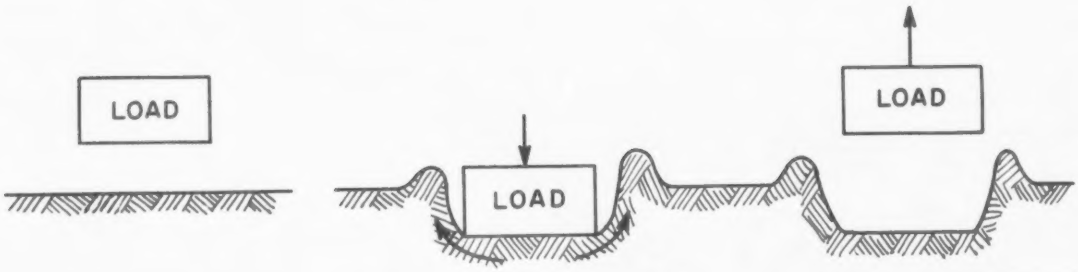
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Figure 80. Plastic deformation.

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for the heaviest vehicle that will use the road. From the manufacturer's specifications, the axle load is divided by the number of tires on the axle to determine wheel load. If dual wheels are on the axle, the wheel load must be increased by 20 percent to account for the overlapping stresses (see Figure 81).

For example, suppose there is an axle loading of 45360 Kg (100,000 lbs) on the rear axle of a truck with dual wheels. It is known that the wheel load is 45360 ÷ 4 or 11340 Kg (25,000 lbs), but because of the dual wheels, one must add 20 percent, or in this case, 2236 Kg (5,000 lbs) to the wheel load to account for the stress overlap under the wheels. The road will now be designed to withstand wheel loads of 13608 Kg (30,000 lbs).

The wheel loading of today's off-highway vehicles is seldom less than 78122.7 Kg per square meter (16,000 lb per square foot), which is the approximate bearing capacity of soft rock. Table 27 shows the approximate bearing capacity of various materials encountered in haulroad construction. As indicated by Table 27 few materials can support large vehicles' wheel loads in their natural state. For this reason, designers must compute the amounts of additional materials that must be placed over the original surface material in order to provide the desired bearing capacity. One of the most widely used methods for achieving this is an empirical method developed by the California Department of Highways in 1942. This is the California Bearing Ratio (CBR) which is the ratio of the load carrying capacity of the test soil to that of crushed rock. The curves shown in Figure 82 were developed for designing road surfaces with known CBR. It should be emphasized that the CBR for any material can be determined by taking a sample of the material to a testing lab. If the CBR is to be assumed, some conservatism should be employed. The use of the chart, Figure 82, is as follows. If the wheel loading is 195306 Kg per square meter (40,000 lb per square foot) and it is desired to build a road over silty clay with a CBR of 4, it can be seen from the chart that the road surface will be required to be 81 cm (32 in) above the original soil (see Figure 83). This will allow the 18144 Kg (40,000 lb) load which is distributed over possibly 0.0929 square meter (1 square foot) at the surface to be distributed over about 3.716 square meter (40 square ft) at a depth of 81 cm (32 inches). A gravel and clay mixture can be used as material with a CBR of 25. From the chart, it is clear that the top of the subgrade must be 25.4 cm (10 inches) from the surface. Crushed stone base with a CBR of 80 can be planned. Crushed stone must be 12.7 cm (5 inches) from the top, as can be seen from the chart.

This leaves 12.7 cm (5 inches) of material still not accounted for between the surfacing and the top of the base. This would normally be filled with a similar base material with a CBR of 80 or some paved surface such as blacktop or concrete.

Materials other than crushed stone for haulroad surfaces have been employed at some mines; however, the advantage of such surfaces is difficult to justify economically. When roads are paved, the surface must be protected when running tracked vehicles such as bulldozers on the road. Spillage is always a problem on haulroads and requires the use of a grader

TABLE 27. BEARING CAPACITIES OF MATERIALS

MATERIAL	BEARING CAPACITY LBS/FT^2
HARD SOUND ROCK	120,000
MEDIUM HARD ROCK	80,000
HARD PAN OVERLYING ROCK	24,000
COMPACT GRAVEL	20,000
SOFT ROCK	16,000
LOOSE AND SANDY GRAVEL	12,000
HARD DRY CLAY	10,000
LOOSE COARSE SAND	8,000
LOOSE FINE SAND	4,000

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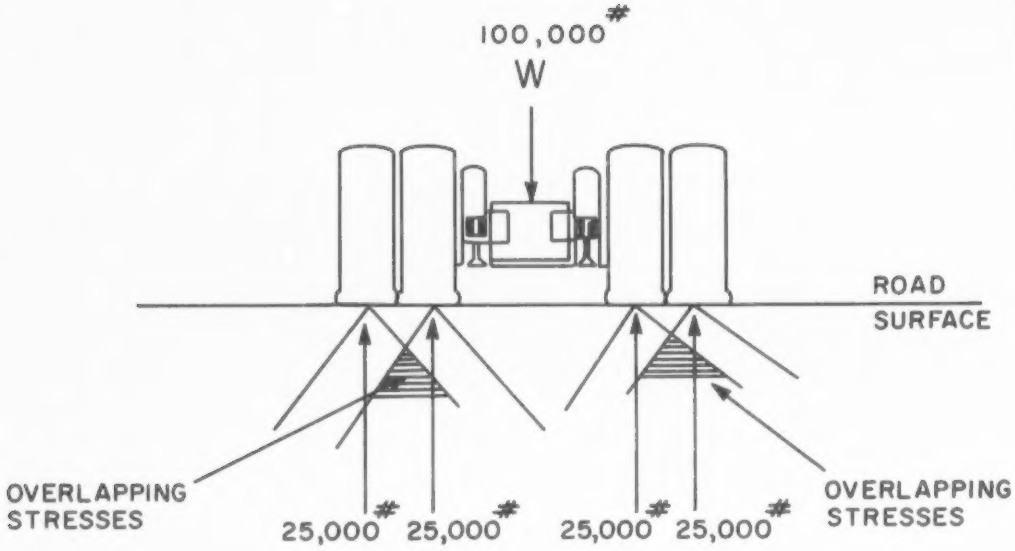
100,000#
W
ROAD SURFACE
OVERLAPPING STRESSES
25,000#
25,000#
25,000#
25,000#
OVERLAPPING STRESSES

Figure 81. Dual wheels on an axle.

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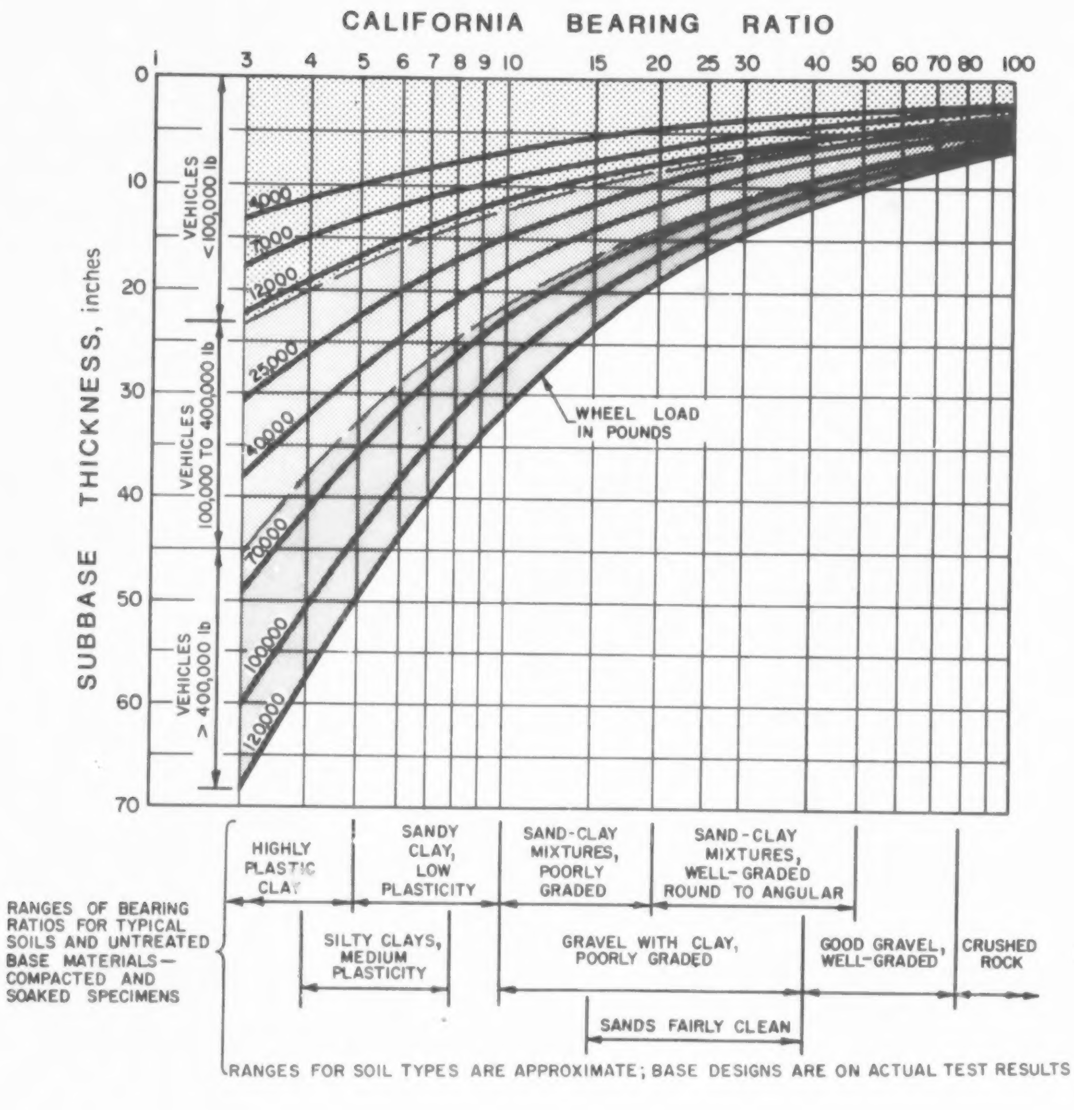

Figure 82. CBR curves.

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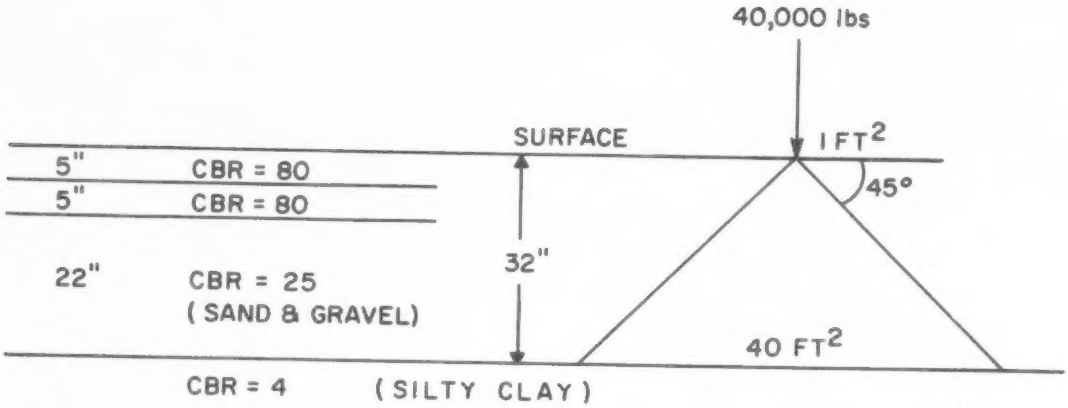
40,000 lbs
SURFACE
1 FT^2
45°
5"
CBR = 80
5"
CBR = 80
32"
22"
CBR = 25
(SAND & GRAVEL)
40 FT^2
CBR = 4
(SILTY CLAY)

Figure 83. Haulroad materials and layer thicknesses.

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for cleaning, even when the road is paved. The grader is used to continually dress and smooth a crushed stone surface so the advantage of paving is unwarranted. The major advantages of surfacing are the reduction of dust during dry weather and the excellent drainage provided during wet weather.

EMBANKMENTS

When designing embankments upon which the load bearing subgrade and base materials are to be placed, it is necessary to understand some characteristics of the various soils to be used. There have been several methods of soil classification of which two are used extensively, namely the American Association of State Highway Officials' (AASHO) method shown in Table 28 and the Unified Soil Classification. The AASHO method seems to be more popular, possibly because it has been used for a longer period of time. There are two other characteristics of cohesive soils that play an important part in describing soils, liquid limit, and plastic limit. When a soil mixture passes from a viscous liquid state to that of a plastic soil, the moisture content at which this transition occurs is called the Liquid Limit (LL). With further reduction in the moisture content the material becomes more stiff and less plastic until it finally reaches a semi-solid state. The moisture content at this point is known as the Plastic Limit (PL). The numerical difference between the LL and the PL is known as the Plasticity Index (PI). Granular soils have a PI = 0. Table 28 shows the various PI for the different classifications.

Any embankment is made up of three components, soil, air, and water. The relationship between moisture content and density is particularly important. All soils have a moisture content at which the soil is at its maximum density. If embankments are placed with the soil moisture content not at this optimum value, the embankment cannot be fully compacted. This optimum moisture content can be determined with a test known as the AASHO density or Proctor density test. Examples of different soil types and their moisture density relationship is shown in Figure 84.

The following tables are included as a guide in selecting embankment material. Table 29 is a selection of soil on the basis of anticipated embankment performance, and Table 30 lists the recommended minimum requirements for compaction of embankments.

When using rock as an embankment material, the size of the individual rock pieces should not be greater than 12 inches in any dimension, and the rock embankment should be placed in layers not to exceed two feet. The top 0.91 m to 1.52 meter (3 to 5 feet) of rock embankments should be placed in soil. The final 30.48 cm (12 in) of embankment should be of material with a dry density of at least 1633.7 Kg/meter cube (102 lb/ft^3) at optimum moisture. When placing earth embankments, the layers should not exceed 22.8 cm (9 inches) loose before compaction for fine-grained cohesive soil and 30.48 cm (12 inches) for granular cohesionless soil.

The final step before actually building the haulroad is to design adequate drainage to keep the road from deteriorating.

TABLE 28. CLASSIFICATION OF HIGHWAY SUBGRADE MATERIALS

General classification	Granular materials (35% or less passing No. 200)						
	A-1			A-2			
Group classification	A-1-a	A-1-b	A-3	A-2-4	A-2-5	A-2-6	A-2-7
Sieve analysis, percentage passing:							
No. 10	50 max.						
No. 40	30 max.	50 max.	51 min.				
No. 200	15 max.	25 max.	10 max.	35 max.	35 max.	35 max.	35 max.
Characteristics of fraction passing No. 40:							
Liquid limit				40 max.	41 min.	40 max.	41 min.
Plasticity index	6 max.		NP	10 max.	10 max.	11 min.	11 min.
Usual types of significant constituent materials	Stone fragments, gravel, and sand		Fine Sand	Silty or clayey gravel and sand			
General rating as subgrade	Excellent to good						

(continued)

TABLE 28. (continued)

General classification	Silt-clay materials (more than 35% passing No. 200)			
				A-7
Group classification	A-4	A-5	A-6	A-7-5, A-7-6
Sieve analysis, percentage passing:				
No. 10				
No. 40				
No. 200	36 min.	36 min.	36 min.	36 min.
Characteristics of fraction passing No. 40:				
Liquid limit	40 max.	41 min.	40 max.	41min.*
Plasticity index	10 max.	10 max.	11 min.	11 min.
Usual types of significant constituent materials	Silty soils		Clayey soils	
General rating as subgrade ..	Fair to poor			

Classification procedure: With required test data available, proceed from left to right on above chart, and correct group will be found by process of elimination. The first group from the left into which the test data will fit is the correct classification.

*Plasticity index of A-7-5 subgroup is equal to or less than LL minus 30. †Plasticity index of A-7-6 subgrade is greater than LL minus 30.

TABLE 29. GENERAL GUIDE TO SELECTION OF SOILS ON BASIS OF ANTICIPATED EMBANKMENT PERFORMANCE

HRB classification	Visual description	Max. dry-weight range, lb per cu ft	Optimum Moisture range, %	Anticipated embankment performance
A-1-a A-1-b	Granular materials	115-142	7-15	Good to excellent
A-2-4 A-2-5 A-2-6 A-2-7	Granular materials with soil	110-135	9-18	Fair to excellent
A-3 A-3a*	Fine sand and sand	110-115	9-15	Fair to good
A-4a* A-4b*	Sandy silts and silts	95-130	10-20	Poor to good
A-5	Elastic silts and clays	85-100	20-35	Unsatisfactory
A-6a* A-6b*	Silt-clay	95-120	10-30	Poor to good
A-7-5	Elastic silty clay	85-100	20-35	Unsatisfactory
A-7-6	Clay	90-115	15-30	Poor to fair

* Ohio modification.

SOURCE: Ohio Department of Highways.

TABLE 30. RECOMMENDED MINIMUM REQUIREMENTS FOR COMPACTION OF EMBANKMENTS

Class of soil (AASHO M 145-49)	Condition of exposure					
	Condition 1 (not subject to inundation)			Condition 2 (subject to periods of inundation)		
	Height of fill, ft	Slope	Compaction (% of AASHO max. D)	Height of fill, ft	Slope	Compaction (% of AASHO max. density)
A-1	Not critical	1-1/2 to 1	95+	Not critical	2 to 1	95
A-3	Not critical	1-1/2 to 1	100+	Not critical	2 to 1	100+
A-2-4	Less than 50	2 to 1	95+	Less than 10	3 to 1	95
A-2-5	Less than 50			10 to 50		95 to 100
A-4	Less than 50	2 to 1	95+	Less than 50	3 to 1	95 to 100
A-5	Less than 50					
A-6	Less than 50	2 to 1	90-95*	Less than 50	3 to 1	95 to 100
A-7						

Remarks: Recommendations for condition 2 depend upon height of fills. Higher fills of the order of 35 to 50 ft should be compacted to 100 per cent, at least for part of fills subject to periods of inundation. Unusual soils which have low resistance to shear deformation should be analyzed by soil-mechanics methods to determine permissible slopes and minimum compaction densities.

*The lower values of minimum requirements will hold only for low fills of the order of 10 to 15 ft or less and for roads not subject to inundation nor carrying large volumes of very heavy loads.

SOURCE: Highway Research Board (4).

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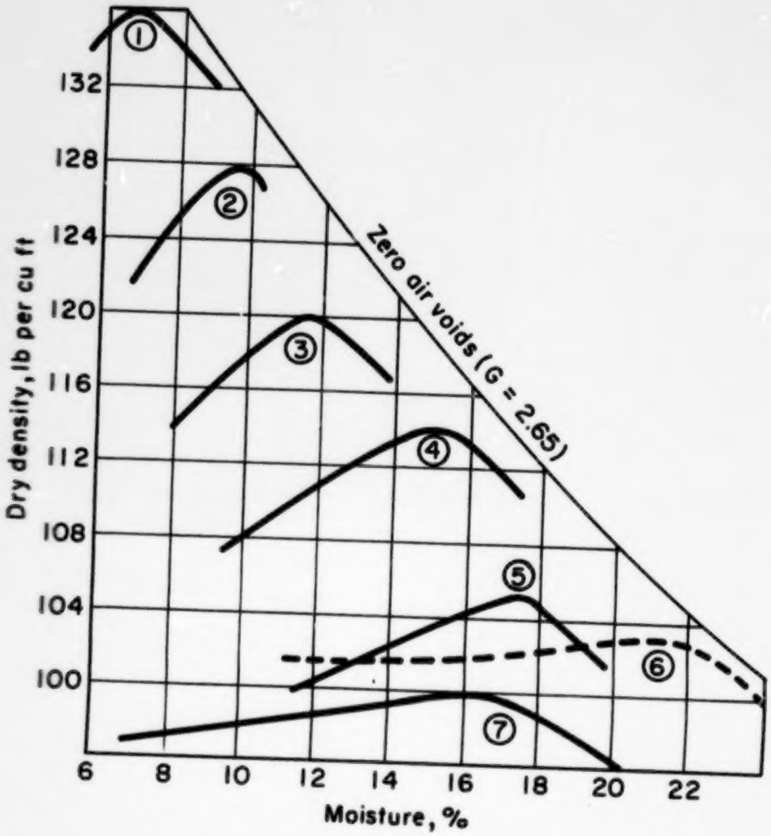

Soil Texture and Plasticity Data

No.	Description	%Sand	%Silt	%Clay	L.L.	P.I.
1	Well-graded loamy sand	88	10	2	16	NP
2	Well-graded sandy loam	72	15	13	16	0
3	Med-graded sandy loam	73	9	18	22	4
4	Lean sandy silty clay	32	33	35	28	9
5	Loessial silt	5	85	10	26	2
6	Heavy clay	6	22	72	67	40
7	Very poorly graded sand	94	← 6 →		NP	NP

Moisture-density relationships for seven soils each compacted by the AASHO standard method

Figure 84. Relationship between moisture content and density.

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DRAINAGE

Once the haulroad has been designed, it is necessary to provide the road with ditches and culverts of sufficient size to remove all surface water before it can damage the road embankments and surface.

If rainfall were applied at a constant rate to an impervious area, the runoff from this area would eventually equal the rate of rainfall. The time required to reach this equilibrium is known as the time of concentration (t_c). This value is the basis for a method of estimating runoff known as the Rational Method. Although there are many methods employed for estimating runoff, this method is simple and does provide reasonable results for small drainage areas which are usually encountered in haulroads. The Rational Method can be reduced to the following empirical formula:

$$Q = K\ i\ A \qquad (1)$$

where

Q = Runoff in cu ft/sec
K = Runoff coefficient, a function of drainage area (Table 31)
i = Intensity of rainfall, in/hour for a duration equal to the time of concentration t_c
A = Drainage area in acres

The time of concentration can be calculated from the following formula:

$$t_c = \frac{60\ (11.9\ L^3)^{0.385}}{H}\ \text{mins} \qquad (2)$$

where

L = Length of channel from headwater of watershed to the end of the channel (miles)
H = The fall in the channel in ft

Once the time of concentration has been established the intensity i can be calculated from the intensity frequency curve shown in Figure 85. This type of curve can be constructed for any location. Each area will have its own unique set of rainfall criteria. The curve shown in Figure 85 is for southern Ohio.

The use of the Rational Method is illustrated for a drainage area adjacent to a haulroad located in southern Ohio (Figure 86). The hatched area of 102 acres will all drain to a new culvert pipe that is to be installed. The values of L, H and K are shown in Figure 86.

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TABLE 31. RUNOFF COEFFICIENTS

Type of Drainage Area	Coefficient of Runoff, K
Concrete or bituminous pavements	0.8-0.9
Gravel roadways, open	0.4-0.6
Bare earth (higher values for steep slopes)	0.2-0.8
Turf meadows	0.1-0.4
Cultivated fields	0.2-0.4
Forested areas	0.1-0.2

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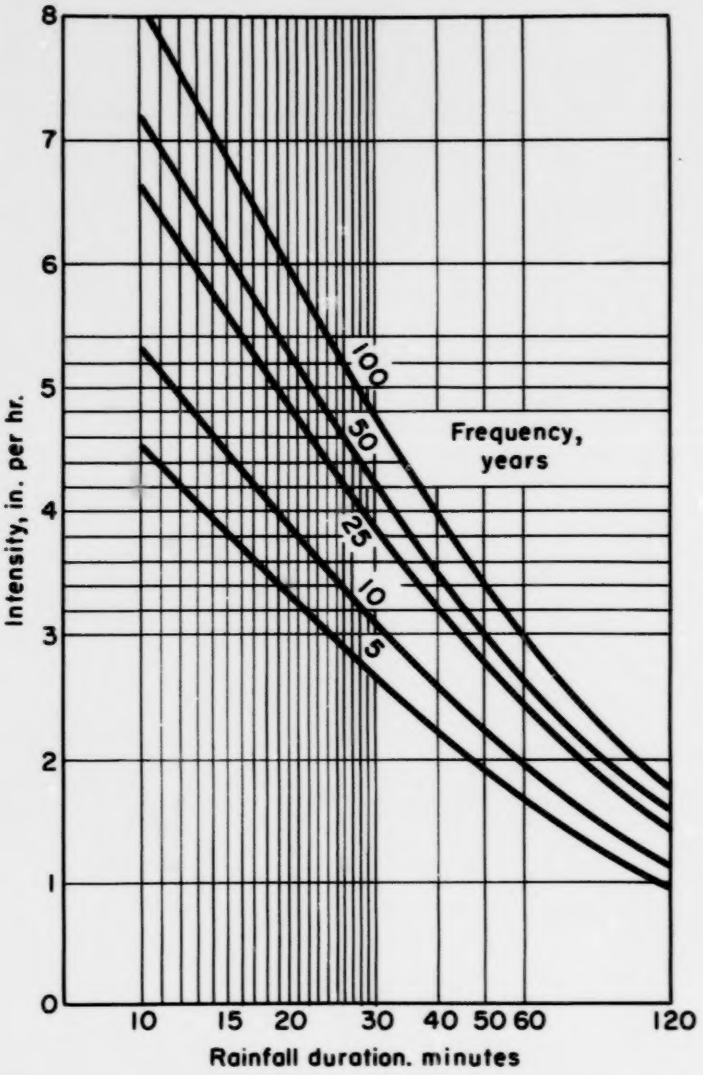

Figure 85. Rainfall intensity, duration, and frequency for the southern one-third of Ohio.

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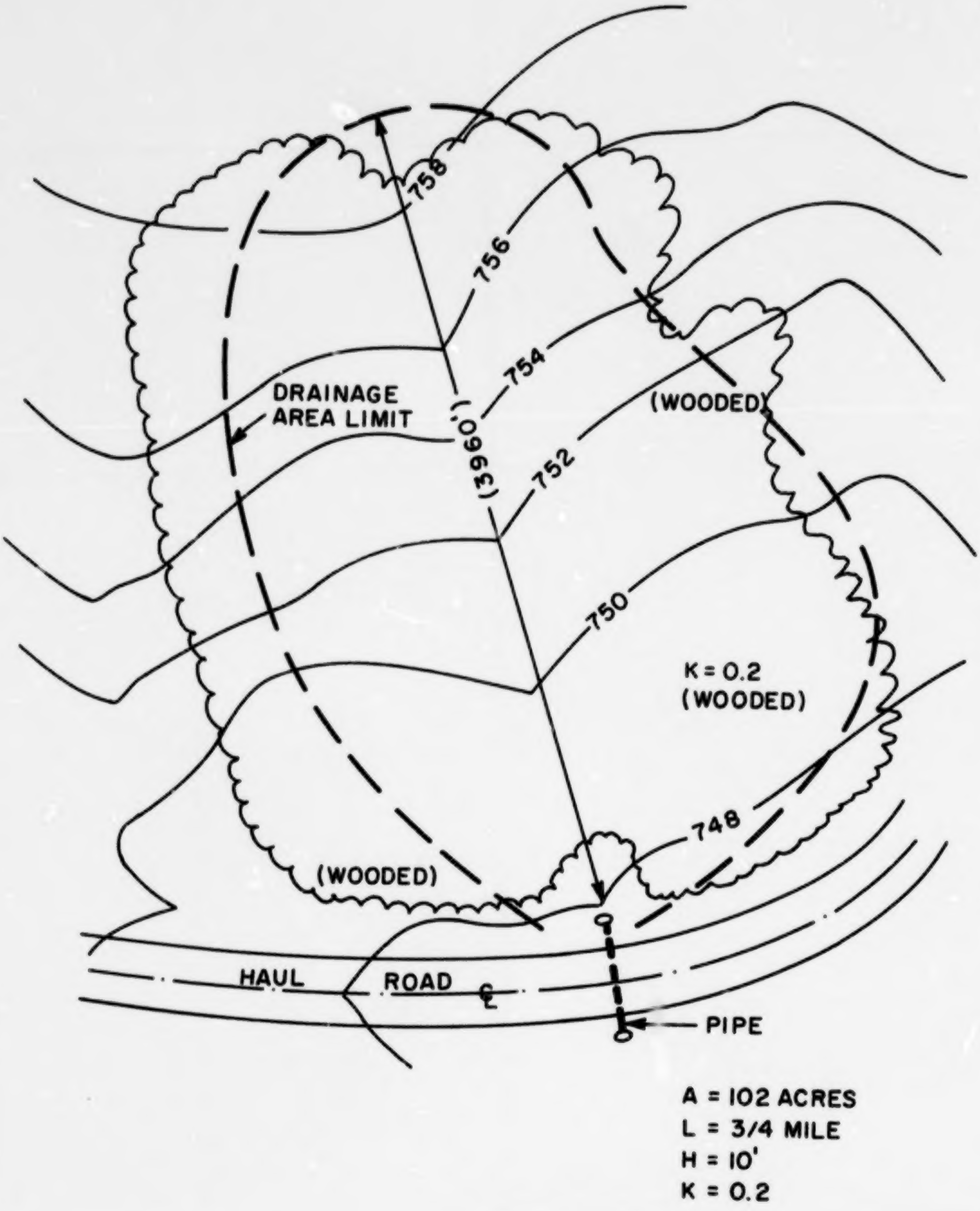
758
756
754
752
750
748
(3960')
DRAINAGE AREA LIMIT
(WOODED)
K = 0.2
(WOODED)
(WOODED)
HAUL ROAD ℄
PIPE
A = 102 ACRES
L = 3/4 MILE
H = 10'
K = 0.2

Figure 86. Drainage area adjacent to a haulroad.

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$$t_c = \frac{60\,(11.9\,L^3)^{0.385}}{H} = \frac{60\,(11.9 \times 0.75^3)^{0.385}}{10} = 46 \text{ mins}$$

From Figure 85, for a 10-year frequency at a t_c of 46 minutes, the value for i is 2.3 inches per hour. Therefore, the value of Q can now be calculated.

$$Q = K\,i\,A$$

$$= 0.2 \times 2.3 \times 102 = 47 \text{ cu ft/sec}$$

There are other more elaborate empirical methods for computing runoff. Regardless of the method, all ditches and culverts must be designed to accommodate the expected runoff.

DITCHES

The use of V ditches for controlling and channeling runoff is economical. As the velocity of water increases with slope, it is advisable to keep all V ditches as flat as possible and yet allow the water to run. Several anti-erosion steps are to be taken within the ditch as the percent slope increases. Rock lined channels are required when slopes exceed 8 percent. The rock lining controls erosion and helps dissipate the velocity energy of the water.

CULVERTS

The size of culverts necessary to take the runoff must be adequate. The most common type of culvert is corregated metal pipe, mainly because of its ease of handling and low initial cost.

When placing pipe culverts in areas of traffic, it is advisable to have a minimum of 60 cm (2 feet) of cover over the pipes; where vehicles of 90718.47 Kg (100 tons) or greater cross the culvert regularly 91 cm (3 feet) would be advisable. When installing pipe culverts the major concern should be to provide lateral confinement to the pipe. The common practice is to excavate a trench 60 cm (2 feet) wider than the outside diameter of the pipe, place the pipe, then backfill in 15 cm (6-inch) maximum compacted layers.

Pipes should never be placed under an embankment prior to placing the embankment. The embankment should be placed first, and then a pipe trench should be excavated through the embankment. The graph shown in Figure 87 shows the size of pipe required for various flow rates. It may sometimes be advisable to place a pipe smaller than that required to take the maximum

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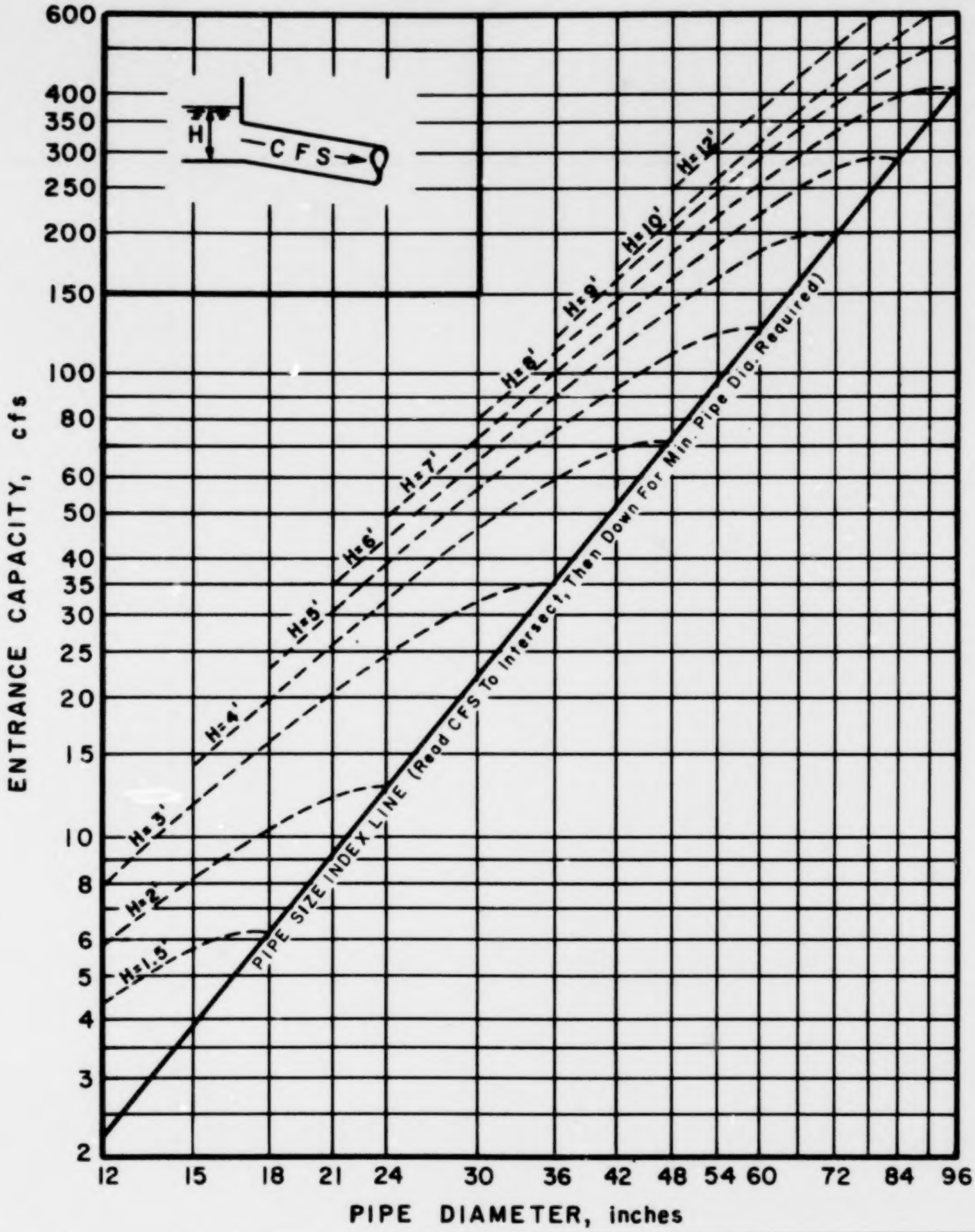

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Figure 87. Pipe culvert capacity graph.

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flow and let the water pond on the upstream side of the culvert. As an example, suppose the flow rate is 0.2832 cu m/sec (10 cu ft/sec). From the graph it can be seen that a pipe larger than 53 cm (21 inches) in diameter will be required. Therefore, a 60 cm (24-inch) diameter pipe is selected. However, if the water could be allowed to pond a maximum of 90 cm (3 ft), one can use a 38 cm (15-in) diameter pipe. Even 60 cm (2 ft) ponding would allow the use of a 45 cm (18-in) diameter pipe.

SUMMARY

This chapter has briefly reviewed the design of surface coal mine haulroads, and the environmental problems associated with them. Good haulroad design for coal mines can hardly be over-emphasized in view of the very heavy loads. Considerations from highway construction, logging roads, etc., can be transferred wherever applicable.

SECTION 7

SLOPE STABILITY

INTRODUCTION

The earth's crust, particularly that portion termed continental, contains a variety of rock types which are arrayed in a wide display of geologic structures and subjected to a great variety of stresses. Some of these structures are very complex, such as overturned and faulted folds, others, such as undisturbed sediments, are very simple. The alteration of ground stability by the introduction of an excavated hole will depend upon the extent of this complexity plus a variety of other factors. The size of the excavation complexity of geologic structure, presence of hydrostatic pressure, and presence of tectonic stresses can all affect the stability of the slopes surrounding an excavation. Luckily, when consideration is given only to coal mines within the regions of interest to this report, the factors simplify themselves greatly. Thus, this slope stability section discusses only the structures of concern to coal mines: highwalls, low-walls, storage piles, and to some extent, impoundments. In this section, the word rock is used generally to cover all crustal material that is found naturally. For the purpose of mechanical analysis, soil is merely rock with a low or absent tensile strength.

The section is divided into three major parts. The first part, the introduction, defines structures of concern, gives a qualitative description of the elements of stability, and ends with a list of helpful references. The middle part, on the theory of stability, gives a quantitative description of the stability elements. The final part of the section gives practical suggestions for constructing a safe slope or for remedying a failing one. The mine engineer should read all three sections, while the operating manager may be content to skip the middle section.

Generally speaking, coal is overlain by sediments possessing relatively simple and flat-lying structure. Although these strata may contain some faults and jointing, the principal mechanical discontinuity is that created by bedding. Since there is but little tectonic disturbance, the stresses in the overburden are primarily due to body weight, that is, the pull of gravity on the rock itself. Therefore, an analysis of slope stability in these coal mines generally requires knowledge of the excavation geometry, rock type and strength, density of bedding, strength across bedding, and the amount and pressure of included groundwater. Many of these data should be determined by careful examination of exploration results, before mining begins.

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Structures of Concern

Man-made slopes in surface coal mines abound; generally, however, they may be classified into four distinct types. Thus, the structures of concern are highwalls, overburden or spoil piles, waste or refuse piles, and impoundments.

Highwalls--As explained elsewhere in this report, highwall heights are usually constrained by economic considerations, with the dimensional limitations of the excavator sometimes acting as a bound. In other words, highwall heights are defined, through stripping ratios, by the price of the coal and the cost of removing it. However, in dragline mining particularly, a maximum digging depth of 50 to 65 m (150 to 200 ft), depending on machine size, is imposed. Rarely does the mechanical stability of the highwall itself enter into these constraints. Generally speaking, this situation exists because coal surface mine slopes are stable with little hydrostatic head in the sediments, and because draglines mine with such speed that they can remove an incipiently failing slope before it becomes catastrophic. However, several specific items enter into the present consideration of highwall stability.

1. No mine is ever totally free from wall failure. Although small, from a mechanical viewpoint, falls of rocks can be disastrous to personnel in the pits. Some major failures have occurred; these can be extremely disruptive to stripping shovels and coal loading machinery working on the pit floor, as well as equipment located on the surface within the shear plane of the failure zone.
2. In times of expensive capital goods, lost time on excavators due to the need to clean up slope failures can be very costly. Human injury and fatality costs are impossible to assess realistically.
3. High-priced coal is leading to ever greater stripping ratios and, hence, higher highwalls. Some wall heights may soon reach the limits of equilibrium.
4. Multiple benching, either to accommodate multiple seams or to allow for more flexible excavation of deep seams, puts large machines on small working platforms. There is limited room available on an intermediate bench to which a machine can be moved if a slope is failing. Although items 1, 2, and 3 refer to limited-height dragline benches, depths of 304.8 m (1000 feet) or more can be achieved by truck-shovel combinations on multiple benches.

Overburden Piles--Spoil piles, often called low-walls, present another version of the stability problem. Mechanically, the material is excavated in discrete particles; it is more homogeneous than when in situ and possesses little or no tensile strength. Since its strength is dependent upon confinement, rather than using a convenient measure of strength such as unconfined compressive strength, more complex measures such as yield criteria and the angle of internal friction are required. Problems in existing mines point to situations which will worsen with increasing depth of mines.

1. Planned stacking space may be inadequate if the swell factor changes at different sites, or if it has been underestimated. To prevent rehandling or covering of coal when this occurs, the temptation is to steepen the slope of the spoil pile. While this steeper slope may be temporarily stable, changes from increased loading or changed weather conditions may cause rapid failure.

2. Mining around an inside curve causes increased amounts of spoil to be deposited in one place. Again, the temptation is to steepen the slope. This is because the fixed casting radius and heights of most stripping machines prevent a safe spreading of the excess material.

3. Maintaining access ramps through spoil causes excess material to be stacked. As with curves, the temptation is to steepen slopes.

4. The presence of water can materially alter a slope's stability. Slopes constructed on a dry day may fail, either by surface erosion or by massive translation, on a wet day. Obviously, the predominant rock type will have a major control on this occurrence. Sandstone blocks, with little recompaction and a great deal of interparticle porosity, will drain quickly and well and remain stable during wet conditions. On the other hand, poorly-cemented sandstones or shales, particularly shales with water-sensitive clays such as bentonite or montmorillonite, may well fail during wet periods.

Storage piles--Although preparation-waste or refuse piles are not peculiar to surface mines, they represent a surface structure which can fail. Inasmuch as their instability may evidence itself years or even decades after the pile was begun, as with the Aberfan disaster in South Wales, great care must be taken in their design and construction. Similar piles that fall into this category are those for the temporary storage of topsoil and for the permanent storage of excess overburden in head-of-hollow or valley fills.

Impoundments--Because of the additional problems created by the storage of water, a separate category is given for those waste piles used as impoundments. The dam at Buffalo Creek, which failed before it was completed, is testimony to the need for consideration of and good design for impoundments.

Elements of Stability

Although a more explicit review of the theory of stability is presented later, a review of the elements of stability is in order in this introductory material. These elements are often grouped, conveniently, into sliding forces and resisting forces; that is those elements which tend to cause failure and those which tend to prevent it.

Sliding Forces--There are two principal sources of sliding forces. The first is the force of gravity on the rock itself or on any equipment resting on the slope; in other words, this force is due to body weight of the slope and the weight of any equipment on top of the slope. The second

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force is that due to any water contained within the material of the slope. Frequently, groundwater pressure is the major contributor to slope failure. A third force, rarely seen in the relatively undisturbed strata in the coal fields, is that of residual stress remaining from mountain-building activity.

Resisting Forces--Those forces that resist failure are unconfined tensile and shear strengths and the shear strength resulting from internal friction. Since the weakest part of an overall slope is often at bedding or jointing planes, the unconfined strengths of concern are those measured across these discontinuities. In addition to its pressure, water can serve to diminish the strength or friction of some rock types. Thus, where clays are present in the rock, water can be doubly damaging to stability.

Sliding Surfaces--The above forces act along pre-existing surfaces such as bedding planes or joints. In slopes of homogeneous material, either solid without pre-existing fractures or granular such as in spoil piles, the forces will act along a curved surface that develops in such a way as to minimize the expenditure of energy during failure. In granular material with no internal friction, this arc will be circular; where friction exists, the arc takes the form of a logarithmic spiral. Apparently, the use of only circles, and not spirals, in the analysis of slopes does not materially affect the prediction of failure. The same cannot be said for the substitution of a plane, particularly when the slopes deviate much from the vertical.

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References

Rather than reference each statement in this section, a list of useful references is given. These references encompass the statements made here, but also provide the design engineer or consultant with a useful library.

"Slope Stability" is discussed in Section 12.1 of Surface Mining [Pfleider, 1968]; essentially the same analysis is presented in Section 7.1 of the SME Mining Engineering Handbook [Cummins, 1973]. Rock Mechanics and the Design of Structures in Rock [Obert and Duvall, 1967] gives background work theory, but has little specific slope stability work.

A recently-released design manual, Engineering and Design Manual: Coal Refuse Disposal Facilities [MESA, undated], is one that no surface-mine designer should be without. It gives a thorough review of slope stability, including extensive references, before embarking on the design of refuse disposal facilities, both piles and impoundments.

Finally, slope stability is discussed in most soil mechanics textbooks. Because the authors did extensive research in stability, a favored reference is Soil Mechanics in Engineering Practice [Terzaghi and Peck, 1967]. One other text, used by this author is Principles of Soil Mechanics [Scott, 1963]. Several other papers, referred to in other parts of this chapter, are also listed.

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THEORY OF STABILITY

Mechanics

The theories of fundamental mechanics are, in their most comprehensive forms, complex and cumbersome. An attempt will be made here to extract only that material needed to understand subsequent analyses.

It is reasonably well understood in mechanics that relationships among the three principal stresses can be expressed which define a material's yield limit. When the stresses are less than the defined relationship, the material is stable. When the stresses reach their limit, or when conditions attempt to push these stresses beyond this limit, the material responds by yielding. This yielding prevents the stresses from exceeding the set limit. At the yield point, further applications of stress on the material result in further deformation but not in a stress rise within the material. Although yield is sometimes called failure, it should not be confused with rupture or collapse. Yield means deformation beyond the elastic limit; the physical manifestation of yield may take a variety of forms from crystal deformation to intergranular slippage to the creation of visible cracks and failure zones. Since failure is energy conservative, microscopic evidences of incipient failure will occur within predictable planes or zones. Further applications of stress will cause the micro-cracks to coalesce into one large crack. Even in granular material, high differential deformation will occur in narrow zones that surround undisturbed blocks. This can be seen in failing slopes, where large blocks will move as one piece along a failure surface.

Although the subject of yield criteria is extensive and yet evolving, particularly for three-dimensional analyses of anisotropic material, the theory for a two-dimensional analysis such as the slip-circle analysis, to be given later, is not that involved. In a general soil, the criterion for stability, and thus failure, was given by Coulomb in the year 1773 as

$$|\tau| \leq \sigma \tan\phi + c \qquad (1)$$

That is, the shearing stress (τ) on a section of isotropic, cohesive soil must be less than the cohesion (c) plus the product of the normal stress (σ) on that same section with the coefficient of internal friction ($\tan\phi$). Failure occurs when the shear stress equals the cohesion plus this value of the normal stress. The geometry of this situation is shown in Figure 88. In soils, cohesion can be thought of as unconfined tensile strength; in rock, however, the tests for shear strength give a better value for cohesion than do those for tensile strength. Since it is difficult, if not impossible to test soils directly for their tensile strength, cohesion is usually calculated from a series of shear tests.

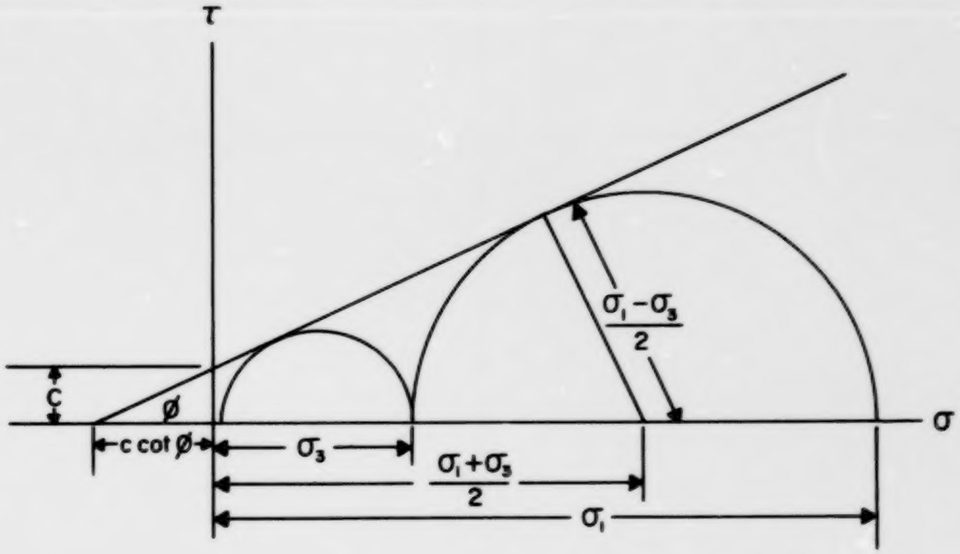

Figure 88. Coulomb's failure surface.

Mohr adapted this yield condition and represented it by means of a graphical plot of principal stresses upon stress co-ordinates, where the normal stress, σ, is the abscissa and the shear stress, τ, is the ordinate. In Figure 88, σ_1, the major, and σ_3, the minor, are the principal stresses, c is the cohesion, c cotϕ is the tensile strength, and ϕ is the angle, called the angle of internal friction, made by the envelope surrounding and tangent to the stress circles and the abscissa. The envelope describes the limiting yield conditions of the principal stresses and satisfies the equality of equation (1); therefore a circle that is tangential to the envelope represents a stressed soil sample that is yielding, and a circle that lies within the envelope represents a stressed soil sample that is rigid. Under the stated conditions, it is impossible for the stresses to be arranged such that a Mohr circle intersects the envelope. In terms of the principal stresses, failure occurs when

$$2c \cos\phi + (\sigma_1 + \sigma_3)\sin\phi = \sigma_1 - \sigma_3. \qquad (2)$$

Considering cohesionless soil, c cosϕ must equal zero and equation (2) reduces to

$$\sigma_1/\sigma_3 = (1 + \sin\phi)/(1 - \sin\phi). \qquad (3)$$

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Equation (3) represents the maximum value of σ_1, for any σ_3, and any ϕ; however, if Figure 88 is examined, it is seen that a circle can be drawn to the left at σ_3 so that σ_1 is smaller than σ_3. This is the minimum value of σ_1 and is represented by

$$\sigma_1/\sigma_3 = (1 - \sin\phi)/(1 + \sin\phi). \qquad (4)$$

The ratios of equations (3) and (4) are recognized as defining Rankine's lateral earth-pressure coefficients. The major principal stress (σ_1) divided by the minor principal stress (σ_3) of equation (3) is called the passive coefficient (K_p) and the same relation in equation (4) is the active coefficient (K_a).

It is known that when the limiting conditions of equilibrium are reached, the soil will fail along preferred orientations. In plane conditions, these orientations are called sliplines and form angles of $1/4\pi \pm 1/2\phi$ with the principal or algebraically greater stress direction. Only in homogeneous and isotropic material can it be shown that stress sliplines are coincident with strain characteristics.

The implication of a positive value for ϕ in a material is that there is an increase in shear strength for each increase in normal stress. Put differently, a confined soil will be stronger than an unconfined soil. The weight of a structure will provide the confinement necessary to create sufficient foundation strength to support that structure, presuming, of course, that yield is not exceeded. A zero value for ϕ, as seen in cohesive materials such as clays and most metals, means that, derived from Equation (1),

$$|\tau| \leq c \qquad (5)$$

and, in terms of Equation (2), failure occurs when

$$|\sigma_1 - \sigma_3| = c \qquad (6)$$

A purely cohesive substance is weakened, not strengthened, by confining pressures. Equation (6) is known as the yield criterion of Tresca, and is often applied in metal plasticity.

Demonstration of the notion of yield criteria and establishment of values for c and ϕ is done with a shear tester. Either a direct-shear (shear box) or triaxial tester may be used. The shear box is preferred when drained tests are to be performed and where strengths across predetermined sliding surfaces are desired. The triaxial tester is preferred for homogeneous material which is allowed to form its own failure plane.

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The triaxial tester can also be arranged to manipulate pore pressure. Thus, the rise of pore pressure due to deformation can be measured or the effect of increased pore pressure (effective stress: see the subsequent section on effect of water) can be simulated. Both tests are standard and can be obtained commercially.

Effect of Geology

In the previous discussion on mechanics, reference is made to homogeneity and isotropy. However, earth materials, especially undisturbed overburden, are unlikely to be uniform. Therefore, some consideration must be given to the effect of geology. Namely, strengths and angles of internal friction vary among the rock types; also, within a single rock type these variables may change according to direction of measurement. Finally, the presence of discontinuities, such as bedding planes, faults, and joints, may reduce unconfined strengths to zero. Since friction exists between any two surfaces, angles of internal friction are often unaffected by discontinuities. This statement is modified by the fact, that many discontinuities, particularly faults, are filled with gouge or other clay-like material which acts as a lubricant and reduces the angle of internal friction.

The effect of geology, therefore, is to take the homogeneous system discussed previously and impose upon it predetermined sliding surfaces and highly variable strength factors. The mine engineer must analyze each slope according to both its geometric and geologic factors. A purely homogeneous and cohesionless material can be analyzed on geometric factors only. For instance, in an undisturbed highwall the sliding surface will be strongly influenced by the strength across bedding planes. If, as is often the case, the contact between the coal and its immediate superjacent stratum (roof) has low or even zero strength, and if the wall fails, it probably will be by sliding (translation) rather than by rotation. Should rotation occur, it will undoubtedly intersect this contact plane. Since coal itself is a weak foundation material, it would be improbable to find a rotational failure which involved the strata below the coal. That is, it is unlikely to find base-circle failures. Similarly, the presence of faults or joints (cleat) will help to define the sliding surface.

Looking closely at the rocks which compose the slope, it is seen that a high degree of variation can occur within a single geologic material. The best example is shale, which may have a shear strength of over 1000 pounds per square inch normal to bedding and a strength near zero parallel to bedding. Similar variations can occur in the angle of internal friction. This angle, ranging between 30 and 40 degrees for most soil materials, is a direct measure of surface roughness and particle shape and is closely correlated to mineralogy. A smooth spherical particle will have a ϕ of around 20 degrees, while an angular substance can have a ϕ as high as 45 degrees. Angles of internal friction over 45 degrees probably represent interlocking particles rather than true friction. Some soil materials, those that contain clay and most notably the shales, can have dramatic alterations to ϕ with the addition of a small percentage of water. Sand and gravel, derived from quartz or unaltered silicate rocks, will demonstrate cohesionless characteristics (c near zero, ϕ between 30 and 40 degrees), while clays and clay stones will demonstrate either cohesionless or cohesive (a measurable c, ϕ near zero) behavior, depending upon the amount of water present. Obviously, shale particles, with little or no cohesive strength, can lose all strength when suddenly wetted.

Narrowing from a general discussion to the specifics of surface coal mines, certain simplifying conditions are seen to exist. Undisturbed strata are flat, and maximum shear strength is parallel to the vertical; as a corollary, maximum tensile strength is normal to the vertical. This is a relatively favorable orientation for slope stability. However, of greatest significance is that most slopes of interest in coal mines are composed of broken materials. Spoil piles, refuse piles, and certainly, the blasted portion of the highwall can be characterized as cohesionless soils. Because the dragline so often sits on blasted material, the last is often of great concern.

Effect of Water

In the preceeding paragraphs, mention was made of some simplifying conditions; it is now appropriate to introduce one important, complicating condition. The presence of water in a slope can reduce its overall stability or even induce catastrophic failure. As discussed in other portions of the background reports, water enters a slope as precipitation, as surface drainage, or as groundwater. Succinctly, water can increase body weight, which is a sliding force, reduce normal stress which contributes to the resisting force, and, for some minerals, reduce ϕ which is tantamount to lubrication and thus a decrease of resisting force. The following comments will give some guidelines on how water affects stability.

Water's Weight--All earth material contains holes or empty spaces termed voids. Even solid rock contains cracks and fissures of interparticular spaces in sediments. When the rock contains a large volume of voids, it is termed porous. When the voids are connected so that a fluid may flow through the rock, it is termed permeable. The ratio of void volume to solid volume is appropriately termed the void ratio. The ratio of void volume to total volume is called porosity. These voids, whether in rock or in soil, contain either fluid or gas. Commonly the fluid is water, but may be petroleum liquids; the gas is usually air, but can be hydrocarbon. For these purposes, concern lies only with air and water. When the entire void space is filled with water, the soil is called saturated. Percent saturation is a ratio of water volume to total void volume.

Similar to rock, soils may be permeable or impermeable, porous or dense, according to their makeup. Soils with particle sizes of sand or larger (greater than one millimeter) are usually porous and permeable. A high degree of fines or clays will decrease a soil's permeability and increase its density. This is because the fines can occupy the void spaces between the larger particles. Naturally, for one kind of particle, the densest soil will be well-graded, that is possessing particle sizes from the finest to the coarsest. Distinction is made between particle density, which is a function solely of mineralogy, and bulk density, which is an overall measure and is connected to both particle density and void ratio.

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Concern lies with the fact that water can add weight to a slope by filling voids. If a rock, whose density is 2.65 grams/cm^3 (165 lbs/ft^3), has a void ratio of 20 percent, then its bulk density is 2.21 grams/cm^3 (138 lbs/ft^3). At saturation, its density is 2.38 grams/cm^3 (148 lbs/ft^3) which is an 8 percent increase. However, 20 percent void ratio equals 17 percent porosity and is quite low for soils. For a soil, with a 50 percent void ratio (33 percent porosity) and the same particle density, bulk density would be only 1.89 grams/cm^3 (118 lbs/ft^3). Saturation gives a bulk density of 2.18 (136) or a percentage increase of 15. Since a soil is much more likely to saturate than intact rock, this weight change should be of concern to exposed spoil piles, refuse banks, and blasted highwalls.

Water pressure--When the voids of a soil become saturated, it is possible for the pore water to develop a pressure which is measurably greater than atmospheric. This pressure is dependent only upon the pore water head and not upon its volume. Hence, even a low porosity material can develop a pore pressure. For simplicity, assume that the water is not flowing and that consideration is given only to the static head. The development of pore-water pressure can lead to the most disastrous situations. These range from improbable and feared quicksand to the collapse of the Buffalo Creek Impoundments on February 26, 1972, which caused an estimated death toll of 118 people.

The net effect of pore pressure is to reduce directly the normal stress equation (1). The notion of effective stress ($\bar{\sigma}$) must be introduced,

$$\bar{\sigma} = \sigma - p_w \tag{7}$$

Equation (7) says that the normal stress on a surface should be the stress due to the body weight of the substance plus any imposed loads, such as the weight of a dragline, less the pore-water pressure (p_w). This reduced quantity is the effective stress. Equation (1) can be rewritten

$$|\tau| \leq \bar{\sigma} \tan\phi + c. \tag{8}$$

The natural and logical question is what happens when $p_w = \sigma$? Obviously, $\bar{\sigma} = 0$, $\bar{\sigma} \tan\phi = 0$ and the substance will behave as a liquid if c is inconsequential. Since liquids have no shear strength, the effect of having a zero value for effective stress is to cause a zero shear strength and to cause seemingly liquid behavior in a bulk solid. This process, called liquefaction, can happen precipitately fast, since saturation and then pressurization of the pores can occur with the addition of only a very small quantity of water. Indeed, during earthquakes, such as at Hebgen Lake, Montana in 1959, where 28 people were killed by a landslide, liquefaction can occur due to the consolidation of the soil from the quake's vibration and the accompanying reduction in porosity, and hence, a rapid rise in pore pressure.

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Mine designers must guard against pore water pressure in all surface mine structures. Areas of concern are joint fractures in a highwall which intersect the surface and can intercept surface run-off; blasted areas of a highwall which collect run-off; fill areas (head-of-hollow or valley) and spoil piles which saturate due to precipitation, surface run-off, or groundwater flow; and, most specifically, impoundment structures and their foundation zones. Water interception structures such as diversion ditches, free-flowing (French) drains where appropriate, and in the extreme, dewatering wells should be built to keep these structures drained. Pore-water pressure can be measured by placing packers and a piezometer in specially drilled wells. Even without precise measurement of pressure, its presence can be presumed if there is a water seep or flow from a slope's surface.

In closing, it should be noted that the so-called lubricating effect of water is often the liquefaction of narrow failure zones. Liquefaction can occur when a porous layer is entrapped by impermeable layers. While this is not often a problem in undisturbed terrain, it immediately becomes so when an excavation enhances the instability of a constrained and pressurized layer. Liquefaction can also occur within fault or joint zones, particularly if they are coincident with failure zones predicted by mechanics theory.

DESIGN OF SAFE SLOPES

The following discussion draws from those preceeding, but can, for ease of reading be taken on its own. It starts with some computational schemes--as it were, theoretical analyses--for factor of safety in a standing slope. It then proceeds into some practical or field considerations for slope stability. The mine designer is warned against taking the simplest approach if it is not appropriate for his conditions. Evaluate the site and select the approach that encompasses the given conditions.

Factor of Safety, First Considerations

Generally speaking, the mechanical stability of any substance can be found if the stresses within that substance and the substance's strength (stresses that the substance can withstand at the point of failure) are known. When failure, in the sense of yield, occurs at a point, such as on an unconfined surface, where strain may develop, then failure may progress to the point of complete mechanical rupture. Because of confinement, it should be remembered that yield failure need not necessarily lead to rupture. By the use of appropriate yield criteria and computers to handle the iterative computations, it is possible to perform such a total stress analysis on a slope.

Because of the complexity of these total solutions, and because of an inability to perform the iterative computations before the widespread introduction of digital computers, a series of limit-equilibrium-analyses were devised. Some of these analyses are still appropriate for the field engineer. Basically, these analyses compare those force components inducing

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a slide against those resisting this slide. To make the problem tractable, a rupture or slip surface has to be assumed. The ratio of resisting forces to sliding forces along any slip surface is the factor of safety for that surface. This is expressed in equation (9),

$$\text{Factor of Safety} = \frac{\Sigma \text{ Resisting Moments}}{\Sigma \text{ Sliding Moments}}, \qquad (9)$$

where the moments are the forces multiplied by the radius. Because slip surfaces are assumed, a series of analyses using differently-located slip surfaces is performed in order to find the minimum factor of safety for any slope.

Sliding forces are usually due to weight and water pressure. The weight of the slope material, its contained water, and any surface loads due to structures or machines, resolved to the components parallel to the slip surface, are the prime cause for sliding. Any unbalanced pore pressure can also add to sliding. Other, more limited, sliding forces might be due to residual ground stresses or formation pressure resulting from expansion of wetted clays or shales.

Resisting forces would be from cohesive strength across the sliding surface and from friction. Obviously, friction forces are a function of the normal component of weight and the coefficient of friction. Resistance may also be due to toe loadings, such as those that occur in base slip circles.

The slip surface itself is often thought to be circular, although other curves such as a logarithmic spiral are possible. Irregular slip surfaces develop where discontinuities such as faults and joints exist in the ground. Often, slip surfaces are a mixture of smooth curves and connecting straight lines. This can be visualized if pre-existing geology is imposed on the theoretical circle of Figure 88. If one of the circles were to intercept a vertical joint which had been opened due to freezing, then the upper part of the slip surface would be defined by the joint rather than a continuation of the arc. Figure 89 can also be used to locate the critical circle (lowest factor of safety) for cohesionless slopes ($\phi = 0$). The variable D is the depth to a firm foundation and is expressed as a function of the slope height (H). Since the coal is assumed to be the solid foundation, most surface coal mine failures will be toe failures and D = H. It should be emphasized that this simplifying condition does not exist for impoundments, refuse piles, and fill areas that are built on soil or poor foundation materials.

Slip-circle Analyses

A calculation of a slope's factor of safety requires some manipulative method to simplify the determination of forces. This calculation is delineated in the paragraphs which follow and the steps to follow are listed

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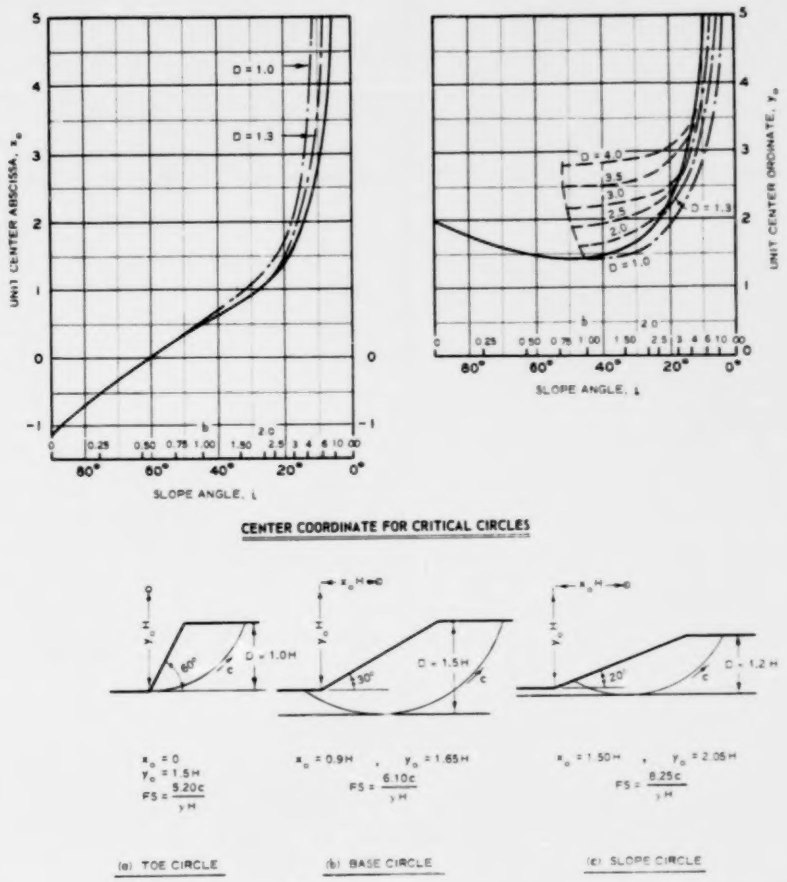

Figure 89. Critical slip circles for purely cohesive slopes, ϕ=0.

in Table 32. Once a rupture surface on which failure will occur is assumed, the slope is divided into slices and each slice is assumed to have the chord, defined by its sides, as its base rather than the intercepted arc. Ten or fifteen such slices give almost as accurate a solution as using the arc itself. Five or six such slices often suffice, for the straight-based slice method gives a more conservative factor of safety than a circle-based slice. The method of slices is shown in Figures 90 and 91. Figure 90 is an overview of the method and represents the two-dimensional plane normal to the slope face. One assumption in this method is that the third dimension has no effect on the analysis. This assumption is called plane strain. Figure 91 shows the force on an individual slice.

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TABLE 32. STEPS IN COMPUTING SLOPE SAFETY FACTORS

1. Draw a true cross-sectional profile of the slope to be analyzed. If the location of bedding, faulting, jointing, or other mechanical discontinuities is known, then place these on the cross-section.

2. Impose a slip circle on the slope. For a first trial, consult Figure 87, and use the following configuration.

   a) Highwall over coal: toe circle.
   b) Overburden or refuse pile on solid foundation: slope circle or toe circle.
   c) Overburden or refuse pile on weak foundations: base circle.
   d) A bench in homogeneous material (highwall and foundation are in the same material): base circle.

3. If groundwater is known to be present, impose the phreatic surface (groundwater table) on the slope.

4. Divide the slope into equal-thickness slices as shown in Figure 90. Determine the weight, including groundwater of each slice. Assign an angle of internal friction and a value of cohesion for each slice base. Be conservative and consider worst-case conditions such as in wet or wintery weather. Measure α for each slice.

5. Determine or assign pore-water pressures, if any, for each slice.

6. Calculate the sliding force according to equations (10) and the resisting force with equation (11) for each slice.

7. Summarize the forces and determine the factor of safety according to equation (12).

8. Repeat this procedure starting with step 2. However, alter the position of the slip circle. At this point, experience has been gained with the analysis, it is wise to try other slope slip surfaces. Instead of a complete circle, for instance, a partial circle truncated by fault or joint cutoffs can be tried. If a plane sliding surface is observed in the slope face, then this should be used also as the base for slices. Continue repeating the analysis until confident that the lowest factor of safety has been calculated.

9. If this factor of safety is less than 1.5 for a working slope, or 4.0 for a permanent slope, then it is necessary to redesign the slope, altering either slope height or overall slope angle or both.

Looking at the idealized slice, it is seen that the sliding force is the component of the total weight, W_t (W_t = weight of solid, W_s + weight of water, W_w + weight of overlying equipment or structures, W_e) which acts parallel to the base or

$$F_s = W_t \sin \alpha, \qquad (10)$$

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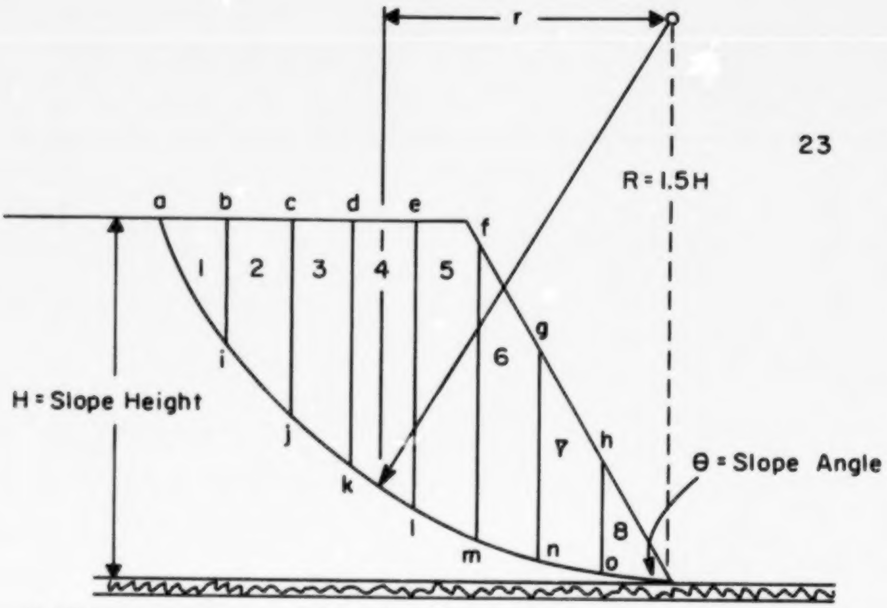
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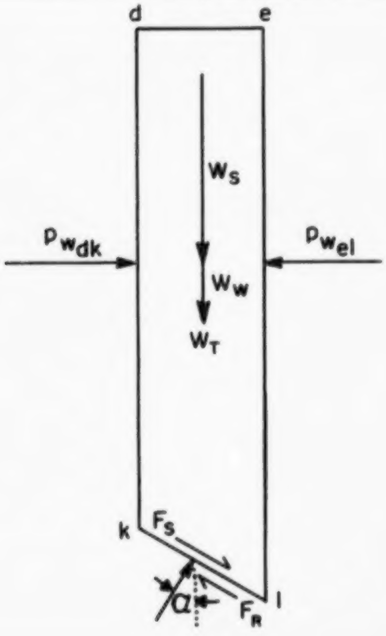
H = Slope Height
θ = Slope Angle

Figure 90. Slices on a toe circle for use in slip-circle analysis of slope stability.


d
e
W_S
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k
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α
F_R

Figure 91. The forces on slice 5.

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where α is the angle between the normal and the vertical. For the purposes of calculation, the slice can be considered to be one foot in depth; this allows the multiplication of the slice area by its assigned density to obtain W_t.

The resisting forces are the combination of shear strength due to friction and cohesion as defined in equation (8). It should be noted that the method of slices compares forces, not stresses, and that the equations in this section refer to the forces on a slice, not stresses (force per unit area). Since the friction or resisting force is a function of the effective normal force on the sliding surface, equation (11) can be written,

$$F_R = (W_t - \overline{Pw}_{kl}\, L_{kl}) \cos\alpha \tan\phi_{kl} + c'L_{kl} \qquad (11)$$

where W_t is the total slice weight, $\overline{Pw}_{kl}$ is the pore-water pressure acting on line kl, L_{kl} is the length of line kl, ϕ_{kl} is the angle of internal friction along kl, and c' is the effective cohesion along the line kl. Effective cohesion refers to the cohesion that exists when wetted. If there is no groundwater and the slope is dry, then c' = c. Substituting in equation (11), using moments instead of forces, gives the analysis for any one circle.

$$\text{Factor of Safety} = \frac{R\Sigma\,(W_t - \overline{Pw}L)\cos\alpha \tan\phi + R\Sigma c'L}{R\Sigma\ W_t \sin\alpha} \qquad (12)$$

An equivalent value for the denominator is

$$\text{Sliding Moments} = \Sigma W_t\, r = R\Sigma\, W_t \sin\alpha .$$

When performing the analysis, it is often helpful to tabularize the variables. It must be remembered that the analysis should be repeated with different circles until a minimum factor of safety is found. Both the circle's radius and the circle's center should be varied. If it is impossible to obtain a factor of safety greater than that required by regulation, then the mine engineer must consider changing the slope itself by altering either H or Θ or both.

Factor of Safety, Additional Considerations

The analysis just presented has several simplifying conditions implicit in it. However, the method of slices will accommodate a more complex approach. For instance, the weight of a slice was given as if the density

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were uniform throughout the slice. It is relatively easy to impose layered geology over Figure 90 and calculate W_t as the aggregation of the weight of each layer present in the slice.

No mention was made of the net effect of pore water pressure on the sides of the slice and its contribution to sliding. Returning to Figure 91, the average pore-water pressure acting on side dk (Pw_{dk}) times the length of dk that is pressurized (the pore-water pressure need not act along the entire side) minus the similar force on side el gives an additional component of sliding. Converting this force to a moment and adding it to the denominator has the effect of lowering the factor of safety. This calculation for side pressure is not precise because it assumes that the side pressures act on the same horizontal plane and do not create a moment couple. A computerized analysis of slices has been developed to handle these unbalanced forces, but its use goes beyond the level of this report.

Additionally, the analysis given assumes that each slice stands alone and shares no strength with its neighboring slice. This assumption is valid for broken material but not for intact geologic material. The more sophisticated computer analysis (Morgenstern and Price, 1965) mentioned above is capable of handling this additional strength variable. The effect of including this side strength is to increase the safety factor so that ignoring it tends to be a conservative move. Finally, the analysis ignores residual ground stresses due to mountain-building activities. It also ignores poro-water pressures due to the movement of groundwater.

The U.S. Bureau of Mines has published a report (Wang et al, 1972), RI 7685, which describes a computer program for slope stability that combines a total stress analysis with a limiting equilibrium analysis. Basically, the program determines the most likely sliding surface by using a finite element stress analysis. It then determines the factor of safety for that surface. For the computer-oriented engineer, this approach gives the most realistic and rapid solution to stability determinations.

It should be noted and emphasized that the method of slices is not synonymous with slip-circle analysis. Although, in the example in Figures 90 and 91, slices were used with a slip circle, the method of slices is equally appropriate for any sliding surface. For a plane surface, such as overburden stacked on the outslope of a contour mine, slice analysis can be done, but forces are used instead of moments. Equally, slices can be used with a sliding surface determined from existing geology rather than with an assumed slip circle. If the presumed sliding surface is based on existing conditions, only one factor-of-safety analysis need be performed. As a precaution, sliding surfaces other than the obvious one should be analyzed to be certain that the lowest factor of safety for the slope is known.

Constructing a Safe Slope

In review, the following elements, when varied, contribute to or diminish slope stability: slope geometry (height and angle); the physical properties of the slope material (weight, angle of internal friction, and

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cohesion); geology (structure and presence of discontinuities); the presence of water (its added weight, pore-water pressure which diminishes shear strength, and the possibility of liquefaction); and the strength of the foundation soil or rock which underlies the slope. Slope stability is obtained by varying those factors in the control of the engineer in response to weaknesses or, if lucky, strengths over which there are no control. Thus, a safe slope is one in which the properties of the slope are known and the slope's height, angle, and method of construction are controlled in response to these properties. The next few paragraphs deal with those elements of stability that can be controlled realistically in coal mines.

Slope Height--In many cases the height of a slope is fixed by premining conditions. However, heights can be reduced by use of top benches or pull back of spoil piles. In multiple-seam mining, it may be necessary to remove an upper seam and its overburden completely in order to limit eventual highwall heights. Concern should be felt when an analysis shows that the factor of safety is below 1.5. Obvious signs of impending failure (see section on monitoring) should lead to immediate corrective action. However, it must be borne in mind that slope stability results from a combination of factors. Since slope height and slope angle both affect stability, it is better, for permanent design changes, to change the slope angle. Nonetheless, the following conditions should be watched.

1. Continuing incipient failure of a dragline bench (tension cracks and slumps) may be an indication that the bench is too high. Since other factors such as dragline productivity through cycle time reduction may be improved, an advance bench may be beneficial. This advance bench, cut by mobile machinery or dragline chopdown, lowers the working highwall.

2. Spoil piles which are allowed to become too high should be watched and repaired. A particular problem is the height of spoil on an inside curve in area mining. Because the dragline continues to dump at relatively the same point as it moves around the curve, the pile builds up above normal height. The excess material should be pulled back by mobile equipment because slope failures would cover valuable coal. The same type of problem exists where a haulroad ramp goes into the pit through the spoil. The spoil at either side of the ramp is higher than normal. Slope failure here could close the haulroad.

3. Refuse piles or overburden storage areas should never be allowed to go above their design height. Regulation may mandate maximum size of these piles; but even if not regulated, their permanence demands a cautious approach to factor of safety.

Slope Angle--Many of the permanent remedies for failing slopes involve reducing the slope's overall angle. Most slope angle reduction methods have the effect of increasing the horizontal dimension of the mine. Thus, a reduction in highwall angle will require an increase in the casting radius of a dragline. A reduction in slope angle of overburden piles, either stacked on a slope or in fills, will require more land area to dispose equal amounts. The following methods elaborate on slope angle reduction.

1. Benches cut into a slope have the effect of reducing a slope's angle. As each bench's width is increased, or as its individual height is reduced, the overall slope angle is decreased. Thus, the use of benches, regularly spaced, is an important way of maintaining stability in deep-haul pits. As stated before, many strip mines use at least one bench, the advance dragline bench, and its design and location should consider stability factors. Multiple-seam coal strip mines give the opportunity to use multiple benches for increased slope control.

2. Even though it is not needed for overall slope reduction, all highwalls which have personnel or vehicles under them should have a safety bench to catch falling rock. Even the most stable slopes will have some loose rock on its face. Loosening can be due to prior blasting or to freezing and weathering. Leaving a safety bench of some five meters (15 feet) in width, no more than one third of the way up the slope, is a good way to safely catch falling rock. In this case, the safety bench is most effective for steep slopes because the falling rocks have little horizontal movement when falling and do not bounce or bound over the safety bench. In the most protected cases, such as public highways, a chain-link fence is provided to catch falling rock.

3. Sometimes a bench is built up at the base of an unstable slope. This transported material increases toe stability, and decreases the overall slope angle. Such built up toes can be used for unstable refuse or overburden fill piles. If the transported material is taken from the top of a slope and placed at its toe, both slope angle and slope height are reduced.

4. The twenty-degree limitation (Public Law 95-87) on out-slope stacking and the current requirement for approximate original contour, have greatly reduced the concern for slope stability of spoil on sloping foundations.

Stabilization of Slope and Foundation--Until now, the physical properties of the slope material have been taken as a given. However, when constructing an overburden or refuse pile, there are many steps that can be taken, other than changing the pile geometry, which enhance the pile's stability. These steps improve the foundation and strengthen the fill.

1. The fill should not go on top of vegetative debris such as brush, shrubs, branches, or general forest matter. Vegetation rots or burns and eventually looses all strength which can create settlement and piping; rotting debris is slippery and can act as a lubricant. Finally, rotting vegetable matter can lower effluent water quality below acceptable limits.

2. Top soil is too valuable to bury with refuse or spoil. Additionally, because of its organic, clay, and water content it makes a poor foundation. Material piled on top soil can fail in the base-circle mode. Top soil should always be stripped from a pile site.

3. Some sub-soil may also be poor foundation material, but yet be too thick to be stripped. If the pile cannot be built elsewhere, some steps

can be taken to improve the bearing capacity of the subsoil. Compaction of the soil will improve it, but maximum compaction occurs only at optimum water content. Dry soils may have to be wetted, or, conversely, wet soils dried. Wetting is by spraying, but drying depends both upon the weather and mechanical exposure (plowing, scarifying, etc.) of the soil to the sun. Proctor compaction tests on the soil with varying degrees of contained water will identify the optimum water content. Sometimes a poor soil can be improved by adding crushed stone. In all cases, good drainage must be provided. When the foundation material is less strong than the pile's, more sophisticated computation of bearing stability should be undertaken than has been described here. A soil mechanics consultant is suggested.

4. It is possible to increase the density and cohesion and decrease the porosity and permeability of cohesive soils and rocks that are put into piles by placing them in horizontal layers and compacting these layers as they are placed. Although compaction will have little effect on purely cohesionless material, most refuse and strip mine overburden has a high proportion of shale, mudstone, or other clay-containing materials. These cohesive materials should be dumped on a horizontal surface and then spread into two-to four-feet thick layers. The layer thickness limitation is so that the weight of the spreading machine will be effective throughout the layer. Layers that are too thick will not be compacted as the stresses from the machine's weight will be too diffuse. High-bearing load spreaders such as rubber-tired dozers or even spreader-compactors should be used in favor of low-bearing machines such as track-type dozers. This form of construction is imperative for refuse piles and strongly suggested for overburden fills where the overburden could be classed as cohesive in nature.

Layering and compacting has decided advantages in reducing spontaneous combustion in refuse piles. It also reduces water infiltration and associated pollution from these piles. Finally, it is a good way of reducing the potential man-made sliding surfaces within a pile. By insuring that all layers are horizontal and by locking one layer to another through compaction, the natural stability of flat-lying sediments is emulated.

Drainage--Generally speaking, a permanently-stable slope is one that does not contain excessive water. Mine slopes, either in the highwall or in the overburden, should be protected from surface water by diversion ditches. Built-up slopes, such as refuse piles or valley fills, especially if they are on sloping ground, should have foundation drains. Existing slopes that show signs of saturation-seepage can be partially dewatered by horizontal wells or drains that are drilled into the base of the slope. These are particularly effective if they intercept vertical fractures or joints. Although good drainage is of paramount importance, it is a complex subject that is handled well in other manuals.

Other Stabilizing Actions--There are a variety of ways to stabilize a slope that have not been discussed. These include cable or steel-bar reinforcement, grouting, retaining walls, and so forth. Most of these methods are expensive and not often used in mining. However, there may be occasions when they are required. For instance, if the slope above a

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major haulroad or other permanent facility, such as a tipple, begins to move, it may be necessary to stabilize it by some complicated means, such as tie rods; there is no room or no time to take the simpler steps that have been described. Again, a consultant should be hired.

For convenience, a variety of slope stabilizing methods are listed in Table 33 (Kimball, undated). Both methods that have been described and those that have only been alluded to are included.

Monitoring

The final portion of this section deals with the monitoring of existing slopes to determine their stability. This monitoring can range from casually collected visual clues to the most intensive observations by scientific equipment. Three categories are examined: visual clues, field surveying, and special slope monitoring equipment.

Visual Clues--Frequently, incipient slope failures are spotted by recognition of surface clues by knowledgeable and responsible mine workers. These people, who should be taught to recognize the signs of impending failure, are encouraged to report their concerns immediately. Optimally, corrective action can be taken to repair the slope; at a minimum, personnel and equipment can be removed from the dangerous area.

Water in a bank is a potential hazard. Therefore water seepage from the face of a bank or from the toe of a bank, unless from a designed drain point, is an early warning sign of problems.

Cracking is a sure sign of slope movement and potential massive failure. Look for tension cracks at the upper part of the slide. If failure is soon to occur, the downhill side of the tension crack will be lower than the uphill side. En echelon shear cracks will appear at the sides of the slumping material, and there will be a zone of uplift or bulging at the base of the slump. The uplift zone, as it spreads, can also show signs of tension cracks. Unlike the crest cracks which are normal to the slide direction, the toe cracks may well be oriented parallel to the slide direction.

If vegetation covers a slope, then these cracks may not be seen readily. However, leaning trees or bare patches of soil, called soil stretching, may indicate movement. Terracettes, which look like regular cattle traces around a hillside, are signs of soil creep. Other indicators of movement are surface features that were installed vertically, or in a row, or both, and that are now out of alignment. Fence lines, stone walls, roads, pole lines, or pipelines that are out of alignment toward the downhill slope may be indications of creep or slumping.

Field Surveying--Actual displacement of sliding can be measured by a variety of field surveying techniques. The precision of measurement is limited only to that of the measurement system itself; there are no peculiarities, in other words, that reduce a method's effectiveness. The

TABLE 33. SUMMARY OF METHODS FOR PREVENTION AND CORRECTION OF LANDSLIDES

Effect on Stability of Landslide	Method of Treatment	General Use		Frequency of Successful Use[1]			Position of Treatment on Landslide[2]	Best Application and Limitations
		Prevention	Correction	Fall	Slide	Flow		
Not affected	I. Advoidance Methods A. Relocation	X	X	2	2	2	Outside slide limits	Most positive method if alternate location economical
Reduces shearing stresses	II. Excavation[3] A. Removal of head	X	X	N	1	N	Top and head	Deep masses of cohesive material
	B. Flattening of slopes	X	X	1	1	1	Above road or structure	Bedrock: also entensive masses of cohesive material
	C. Benching of slopes	X	X	1	1	1	Above road or structure	where little material is removed at toe
	D. Removal of all unstable material	X	X	2	2	2	Entire slide	Relatively small shallow masses of moving material
Reduces shearing stresses and increases shear resistance	III. Drainage: A. Surface: 1. Surface ditches	X	X	1	1	1	Above crown	Essential for all types
	2. Slope	X	X	3	3	3	Surface of moving mass	Rock facing or previous blanket to control seepage

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TABLE 33. (continued)

Effect on Stability of Landslide	Method of Treatment	General Use		Frequency of Successful Use[1]			Position of Treatment on Landslide[2]	Best Application and Limitations
		Prevention	Correction	Fall	Slide	Flow		
Reduces shearing stresses and increases shear resistance (cont'd)	3. Regrading surface	X	X	1	1	1	Surface of moving mass	Beneficial for types
	4. Sealing cracks	X	X	2	2	2	Entire, crown to toe	Beneficial for all types
	5. Sealing joint planes and fissures	X	X	3	3	N	Entire, crown to toe	Applicable to rock formation
	B. Subdrainage							
	1. Horizontal drains	X	X	N	2	2	Located to intercept and remove subsurface water	Deep extensive soil mass where ground water exists
	2. Drainage trenches	X	X	N	1	3		Relatively shallow soil mass with groundwater present
	3. Tunnels	X	X	N	3	N		

(continued)

TABLE 33. (continued)

Effect on Stability of Landslide	Method of Treatment	General Use		Frequency of Successful Use[1]			Position of Treatment on Landslide[2]	Best Application and Limitations
		Prevention	Correction	Fall	Slide	Flow		
Increases shearing resistance	IV. Restraining Structures:							
	A. Buttresses at foot							
	1. Rock fill	X	X	N	1	1	Toe and foot	Bedrock or firm soil at reasonable depth
	2. Earth fill	X	X	N	1	1	Toe and foot	Counterweight at toe provides additional resistance
	B. Cribs or retraining walls	X	X	3	3	3	Foot	Relatively small moving mass or where removal of support is negligible
	C. Piling:							
	1. Fixed at slip surface		X	N	3	N	Foot	Shearing resistance at slip surface increased by force required to shear or bend piles

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TABLE 33. (continued)

Effect on Stability of Landslide	Method of Treatment	General Use		Frequency of Successful Use[1]			Position of Treatment on Landslide[2]	Best Application and Limitations
		Prevention	Correction	Fall	Slide	Flow		
Increases shearing resistance (cont'd)	2. Not fixed at slip surface		X	N	3	N	Foot	
	D. Dowels in rock	X	X	3	3	N	Above road or structure	Rock layers fixed together with dowels
	E. Tie-rodding slopes	X	X	3	3	N	Above road or structure	Weak slope retained by barrier which in turn is anchored to solid formation
Primarily increases shearing resistance	V. Miscellaneous Methods A. Hardening of slide mass: 1. Cementation or chemical treatment							
	(a) At foot		X	3	3	3	Toe and Foot	Non-cohesive soils
	(b) Entire slide mass		X	N	3	N	Entire slide mass	Non-cohesive soils to prevent movement

(continued)

TABLE 33. (continued)

Effect on Stability of Landslide	Method of Treatment	General Use		Frequency of Successful Use[1]			Position of Treatment on Landslide[2]	Best Application and Limitations
		Prevention	Correction	Fall	Slide	Flow		
Primarily increases shearing resistance (cont'd)								temporarily in relatively large moving mass
	2. Freezing	X		N	3	3	Entire	
	3. Electorosmosis	X		N	3	3	Entire	Effects hardening of soil by reducing moisture content
	B. Blasting		X	N	3	3	Lower half of landslide	Relatively shallow cohesive mass underlain by bedrock. Slip surface disrupted; blasting may also permit water to drain out of slide mass
	C. Partial removal of slide at toe			N	N	N	Foot and Toe	Temporary expedient only; usually decreases stability of slide

1 1 = frequently; 3 = rarely; N = not considered applicable

2 Relative to moving or potentially moving mass. 3 Exclusive of drainage methods.

selection of any particular method will depend upon the conditions on the slope and the equipment available to the engineer. Some of the more useful methods are listed.

1. The vertical descent of permanent markers can be obtained by running a line of levels from a stable zone across the slump to the next stable zone. Short lengths of pipe or concrete are sunk into the soil; periodic checks on their level gives their vertical movement. This survey is quick, cheap, and possible with normal mine equipment. It does require a man to cross the slump zone and does not give horizontal displacement.

2. Several surveying methods which use a transit can improve leveling. Again, permanent markers are installed. To get total displacement, vertical angles are taken as well as horizontal angles from a measured base line. Although triangulation computation is more complicated, nobody is required to traverse the slump. Should the slump be stable enough to cross, however, then stadia measurements can reduce computational work. Although a high-order transit traverse can be done across a slump, triangulation is preferred.

3. Electronic Distance Measurers (EDMs), particularly those that embody a transit for angular measurement, substitute well for transits and expedite the survey. However, the targets are not disposable, and thus, a worker is required to cross the zone. Modern EDM's are reliable, replicable, fast and easy to use. They are, however, relatively expensive and subject to deterioration if not used often.

4. Photogrammetry is an excellent way to observe ground motion. The camera records the position of permanent markers; displacements are scaled from the photograph by measuring distances between markers in the slump and one on stable ground. Although aerial photography gives horizontal displacements more precisely than vertical, oblique stereo photography will give both. Actually, photogrammetry does not require an airplane and good results can be obtained with a tripod-mounted camera.

Special Monitoring Equipment--If more sophisticated (although not necessarily more accurate) measurement of slope failure is desired, there is a variety of instruments capable of giving displacements, strains, tilts, or record of acoustic emissions. Home-made extensometers and the most sophisticated telemetric monitors have been used on moving slopes. Some of the reasons for using this equipment over field surveying are continuous output, unattended performance, information from depth, or scientific faddishness. A slope stability consultant will be glad to advise whether any of this equipment is appropriate. Frankly, because much of it is fragile and prone to failure from tampering, it should be used only when normal surveying appears to be inadequate.

Assessing Displacement Information--To close this section, it is necessary to suggest what can be done with monitoring information, without requiring a textbook on rock mechanics. It is suggested that one step is most important. Displacement data should be plotted against time so that total displacement from the beginning of measurement and incremental

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displacements can be observed. The mathematical slope of such a plot is the displacement rate. If the rate is increasing over time, then the mine slope is surely failing. If the rate is decreasing—the graph's slope is flattening—, then the mine slope is stabilizing.

By reasonable extrapolation of a displacement-time plot, predictions can be made of when a slope will fail disastrously. Although continuing but small displacements can be considered a failure, they are of no consequence to a working slope that may soon be removed or stabilized. On the other hand, continuing displacements in a permanent slope are signs of need for immediate, corrective action. If these small displacements get larger day by day, then a day will come when the displacement is total.

SUMMARY

This chapter has given a three-part analysis of slope stability. The introductory and concluding sections can be read independently to get a descriptive review of slope stability elements and the diagnosis of slope failure. If engineering calculations of safety factors are required, then the middle section should also be read. For the engineer who is deeply involved with stability analyses, further references are suggested.

SECTION 8

OPERATING CONSIDERATIONS

INTRODUCTION

In judging the performance of stripping equipment, several material and equipment characteristics need to be taken into account. Given the best of conditions, a piece of equipment may not achieve the optimum level of production due to operational conditions. In practice, the mechanical efficiency of machines never equals one hundred percent, and designed production is rarely realized. Therefore, several correction factors must be applied to adjust equipment performance from a theoretical maximum to that realizable in practice.

CORRECTION FACTORS

The correction factors arise from machine design, material characteristics, and modes of operation.

Swing Factor

The superstructure on which the dragline and shovel revolve can swing 360° in either direction. The cycle time of the machines is affected by the swing angles. The factor that determines the effect of swing angle variations on cycle time, with 90° taken as the normal, is called the swing factor. The smaller the angle of swing, the less the cycle time and the more cycles of operation completed per hour. For example, an angular increase from 90° to 120° for a shovel will increase the cycle time approximately 10 percent (Tables 34 and 35).

Depth Factor

There is an optimum cutting depth for a dragline or a shovel. When the equipment operates at any depth other than this optimum, machine production decreases. The correction for operating at different depths is usually made in the cycle time calculation. Tables 34 and 35 incorporate the effect of depth.

Operating Time Per Hour (MPH)

This factor is applied to derive the machine working time from the theoretical 60 minutes per hour. For example, if a machine works 48 minutes every hour, the MPH factor is 0.80 (48/60) (Table 36).

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TABLE 34. EFFECT OF DEPTH OF CUT AND ANGLE OF SWING ON SHOVEL CYCLE TIME (DREVDAHL, 1961)

Depth of cut in % of Optimum	Angle of Swing in Degrees						
	45°	60°	75°	90°	120°	150°	180°
40	.93	.89	.85	.80	.72	.65	.59
60	1.10	1.03	.96	.91	.81	.73	.66
80	1.22	1.12	1.04	.98	.86	.77	.69
100	1.26	1.16	1.07	1.00	.88	.79	.71
120	1.20	1.11	1.03	.97	.86	.77	.70
140	1.12	1.04	.97	.91	.81	.73	.66
160	1.03	.96	.90	.85	.75	.67	.62

Production at a given depth and angle of swing = Production at optimum depth and 90° swing x Factor for swing and depth

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TABLE 35. EFFECT OF DEPTH OF CUT AND ANGLE OF SWING ON DRAGLINE CYCLE TIME

Depth of cut in % of Optimum	Angle of Swing in Degrees							
	30°	45°	60°	75°	90°	120°	150°	180°
20	1.06	.99	.94	.90	.87	.81	.75	.70
40	1.17	1.08	1.02	.97	.93	.85	.78	.72
60	1.24	1.13	1.06	1.01	.97	.88	.80	.74
80	1.29	1.17	1.09	1.04	.99	.90	.82	.76
100	1.32	1.19	1.11	1.05	1.00	.91	.83	.77
120	1.29	1.17	1.09	1.03	.985	.90	.82	.76
140	1.25	1.14	1.06	1.00	.96	.88	.81	.75
160	1.20	1.10	1.02	.97	.93	.85	.79	.73
180	1.15	1.05	.98	.94	.90	.82	.76	.71
200	1.10	1.00	.94	.90	.87	.79	.73	.69

Production at a given depth and angle of swing	=	Production at optimum depth and 90° swing	x	Factor for swing and depth

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TABLE 36. MINUTES OF OPERATING TIME PER HOUR AND EQUIVALENT OPERATING EFFICIENCIES

Minutes per hour	Efficiencies, %
60	100
55	92
50	83
45	75
40	67
35	58
30	50
25	42
20	33
15	25

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Bucket-Fill Factor

The bucket-fill factor is sometimes referred to as carry factor or dipper factor for shovels and bucket factor for draglines. This factor expresses the actual quantity carried as a fraction of the nominal dipper or bucket capacity, and is a function of the digging conditions (Table 37). These conditions are classified as easy, medium, or hard digging. Since the values describing the conditions are relative, detailed field tests should be conducted to determine the fill factors.

Load and Swell Factors

The load factor (LF) relates the bank volume to the loose volume handled. This factor is particularly important in contracted overburden removal, since payments are based on either bank or loose volume. Mathematically, the load factor is equal to loose density divided by bank density (Table 38). It is expressed as a fraction. On the other hand, the swell factor (SF) is the percentage increase in volume of material from bank state to loose state. The relationship between load factor and swell factor is as follows:

$$LF = \left(\frac{100}{100 + SF}\right)$$

Other Factors

There are several other factors such as management and operator efficiencies and climatic conditions that affect equipment performance. These cannot be accurately quantified. Atkinson (1971) has divided both the job conditions and management practices into four categories each, from excellent to poor and has provided a matrix of values for various combinations of job conditions and management conditions (Table 39). Drevdahl (1961) has also presented a similar table of correction factors for combined job-management conditions.

For some earth-moving equipment, a more detailed breakdown of these factors is possible. For example, Caterpillar Tractor Company has provided correction factors for dozers for grade, visibility, type of materials, and mode of operation. Since values for these factors are specific to particular equipment, they are discussed and presented in the report along with the equipment production calculations.

CYCLE TIME

The cycle time of stripping equipment consists of two distinct components, the fixed and the variable. For a stripping shovel, crowding, dump, digging, and retract times are constant or fixed in every cycle.

TABLE 37. FILL FACTOR (DREVDAHL, 1961)

Easy Digging	Medium Digging
Shovel Dipper Factor 85% to 100%	Shovel Dipper Factor 80% to 90%
Loose, soft, free running materials Close lying, which will fill dipper or bucket to capacity and frequently provide heaped load. Overload compensates for swell of material	Harder materials that are not difficult to dig without blasting, but break up with bulkiness causing voids in dipper or bucket.
Dry sand or small gravel. Moist sand or small gravel. Loam. Loose earth. Muck. Sandy clay. Loose clay gravel. Cinders or ashes. Bituminous coal. Very well blasted material.	Clay - wet or dry. Coarse gravel. Clay gravel, packed. Packed earth. Anthracite coal.

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TABLE 37. (continued)

Hard Digging	Rock
Shovel Dipper Factor 70% to 80%	Shovel Dipper Factor 40% to 60%
Materials requiring some breaking up by light blasting or shaking. More bulky and somewhat hard to penetrate, causing voids in dipper or bucket.	Blasted rock, hardpan and other bulky materials, which cause considerable voids in dipper or bucket and are difficult to penetrate.
Well broken limestone, sand rock and other blasted rocks. Blasted shale. Ore formations (not of rock character) requiring some blasting. Heavy wet, sticky clay. Gravel with large boulders. Heavy, wet gumbo. Cemented gravel.	Hard tough shale. Limestone. Trap rock. Granite. Sandstone. Taconite. Conglomerate. Caliche rock. Any of these blasted to large pieces mixed with fines and dirt. Tough, rubbery clay that shaves from bank.

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TABLE 38. MATERIAL WEIGHTS, PERCENTAGE, SWELL AND LOAD FACTORS (CATERPILLAR TRACTOR CO., 1975)

Weights shown are averages, rounded off to the nearest 50 pounds. Since actual weights are affected by moisture content, grain size, degree of compaction and other local conditions, tests must be made to determine exact material characteristics. The load factor and swell have been determined from the weights given, using the equations found on Page 209.

Material	lb/bcy	% Swell	lb/lcy	Load Factor
Bauxite	3200	33	2400	.75
Callche	3800	82	2100	.55
Cinders	1450	52	950	.66
Carnotite, Uranium Ore	3700	35	2750	.74
Clay - Natural Bed	3400	22	2800	.82
Dry excavated	3100	23	2500	.81
Wet excavated	3500	25	2800	.80
Clay & Gravel: Dry	2800	41	2000	.71
Wet	3100	11	2800	.90
Coal: Anthracite, raw	2700	35	2000	.74
washed	2500	35	1850	.74
Bituminous, raw	2150	35	1600	.74
washed	1900	35	1400	.74
Decomposed Rock:				
75% Rock 25% Earth	4700	43	3300	.70
50% Rock 50% Earth	3850	33	2900	.75
25% Rock 75% Earth	3300	25	2650	.80
Earth - Dry packed	3200	25	2550	.80
Wet excavated	3400	27	2700	.79
Loam	2600	23	2100	.81
Granite - Broken	4600	64	2800	.61
Gravel - Pitrun	3650	12	3250	.89
Dry	2850	12	2550	.89
Dry 1/4"-2" (6-51 mm)	3200	12	2850	.89
Wet 1/4"-2" (6-51 mm)	3800	12	3400	.89
Sand & Clay - Loose	3400	27	2700	.79
Compacted	—	--	4050	--
Gypsum - Broken	5350	75	3050	.57
Crushed	4700	75	2700	.57
Hematite, Iron ore	4900	18	4150	.85
Limestone - Broken	4400	69	2600	.59
Magnetite, Iron ore	5500	18	4700	.85
Pyrite, Iron ore	5100	18	4350	.85
Sandstone	4250	67	2550	.60
Sand - Dry, loose	2700	12	2400	.89
Damp	3200	12	2850	.89
Wet	3500	12	3100	.89
Sand & Gravel - Dry	3250	12	2900	.89
Wet	3750	10	3400	.91
Slag-Broken	4950	67	2950	.60
Stone - Crushed	4500	67	2700	.60
Taconite	7100-9450	75-72	4100-5400	.57-58
Top Soil	2300	43	1600	.70
Traprock - Broken	4400	49	2950	.67

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TABLE 39. OPERATING EFFICIENCY FACTORS (ATKINSON, 1971)

JOB CONDITIONS ↓	MANAGEMENT CONDITIONS EXCELLENT	GOOD	FAIR	POOR
Excellent	0.83	0.80	0.77	0.77
Good	0.76	0.73	0.70	0.64
Fair	0.72	0.69	0.66	0.60
Poor	0.63	0.61	0.59	0.54

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However, the swing and swing-back times can vary from cycle to cycle, and therefore, constitute the variable component of the cycle time. The relationship between production and cycle time is shown in Table 40.

Calculation of theoretical cycle times is rather involved. Manufacturers usually provide enough horsepower for various motions to maintain a uniform cycle time for comparable conditions. Every activity and motion of the machine is dependent on motor horsepower, machine weight, and the need for proper control. For example, when calculating the swingtime, one has to know the average translational inertia of the machine and the average torque from the speed-torque curve. Here, one must also take into consideration the swing-gear ratio. The acceleration and deceleration times, plus the continuous running time at full speed are also taken into account for this calculation. In industry literature, on the basis of machine parameters and design information, standard cycle times have been provided (Table 41). These ideal cycle times are subject to correction for any nonstandard application (chopping) or differences in swing angle, digging height, etc. The influence of these factors were discussed earlier. Cycle times can also be obtained through time studies in the field.

Decreases in cycle time can be achieved by increasing the motor horsepower. For example, maximizing the swing efficiency would not only require an increase in power, but also a reduction in inertia loads, resulting in only a modest reduction in cycle time. Swing power can be increased, but this does not yield a proportional reduction in cycle time. An increase of 50 percent in power, for example, reduces total cycle time of a dragline by only 10 percent. Such major increases in power will be reflected in high operating costs.

For draglines, the swing time is the most critical, although in some cases, hoisting time is important. In cases where the hoisting time makes up to about 40 percent of the total cycle time, an increase of 33 percent in power reduces the cycle time by 10 percent. This reduction in cycle time occurs only in the hoisting time portion of the cycle, unlike the swing-time reduction which spreads over the entire range of the total cycle. The average reduction in cycle time, due to decreases in hoisting time, in actuality, will be less than 10 percent.

The bucket-fill time appears fixed and is not affected by the size of the bucket, provided the machine operator is efficient.

It should be noted that a chopping operation increases the cycle time by as much as 40 to 50 percent. An average increase of 45 percent would be appropriate for calculation over the normal overall cycle time without extended swing or hoisting time (Table 42). Also, bench digging or extension increases the dragline cycle time by as much as 80 percent over the overall cycle time.

Operating Delay

In production calculations, not all the hourly minutes are parts of the cycles as there are inevitable delays that occur which take some of the

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TABLE 40. WALKING DRAGLINE MACHINE OUTPUT (LEARMONT, 1975)

$$\text{Bank Yards per Operating Hours} = \frac{3600}{\text{Cycle Time}} \times \text{Operating Delay Factor} \times \frac{\text{Efficiency Factor}}{\text{Swell Factor}} \times \text{Bucket Capacity}$$

	2570-W 335 Ft Boom, 38°300' Radius, 110 Cu Yd	2570-W 360 Ft Boom, 31-1/2°340' Radius, 90 Cu Yd
Case 1, 110 Ft Overburden		
Weighted Average Cycle Time, From Table 42	48.2	45.7
Operating Delay Factor, From Table 43	0.80	0.82
Efficiency Factor	0.85	0.85
Swell Factor	1.25	1.25
Bank Yards per Operating Hour	40.6 x Bucket Capacity = 4470 yd per hr	43.9 x Bucket Capacity = 3950 yd per hr
Case 2, 130 Ft Overburden		
Weighted Average Cycle Time, From Table 42	57.7	47.7
Operating Delay Factor	0.82	0.84
Efficiency Factor	0.85 Bank Material 0.95 Rehandled Material	0.85
Average Efficiency Factor	0.88	
Swell Factor	1.25	1.25
Bank Yards per Operating Hour (including 38% rehandle)	36.0 x Bucket Capacity = 3960 yd per hr	43.1 x Bucket Capacity = 3880 yd per hr
Productive Bank Yards per Operating Hour	26.1 x Bucket Capacity = 2870 yd per hr	

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TABLE 41. APPROXIMATE CYCLES PER HOUR FOR STRIPPING SHOVELS AND DRAGLINES (ATKINSON, 1971)

Bucket or Dipper Size		Dragline	Shovel
Yd^3	m^3		
8-35	6-27	58	69
36-59	28-45	56	68
60-200	46-150	53	64

* These figures are based on a 90° swing for a shovel and a 120° swing for a dragline, which approximates most field conditions.

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TABLE 42. WALKING DRAGLINE CYCLE TIME CALCULATION (LEARMONT, 1975)

	2570-W 335-Ft Boom, 38° 300' Radius 110 Cu Yd			2570-W 360-Ft Boom, 31-1/2° 340'Radius, 90 Cu Yd		
	Chop Down	Median Depth	Maximum Depth	Chop Down	Median Depth	Maximum Depth
Case 1, 110 Ft Overburden						
Depth, Ft	20	45	90	20	45	90
Percent of Total	18	41	41	18	41	41
Average Swing Angle, Degree	120	45	90	105	40	80
Spot Bucket, Sec	3.0	3.0	3.0	3.0	3.0	3.0
Fill Bucket, Sec	10.2	10.2	10.2	9.5	9.5	9.5
Hoist Bucket, Sec	--	--	20.6	--	--	--
Swing Over, Sec	23.2	13.5	--	22.0	13.2	18.8
Swing Back, Sec	22.3	12.7	18.5	21.0	12.4	17.8
Total Time, Sec	58.7	39.4	52.3	55.5	38.1	49.1
Weighted Average Cycle Time, Sec		48.2			45.7	

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TABLE 42. (continued)

	2570-W 335-Ft Boom, 38° 300' Radius 110 Cu Yd				2570-W 360-Ft Boom, 31-1/2° 340' Radius, 90 Cu Yd		
	Chop Down	Median Depth	Maximum Depth	Extend Bench	Chop Down	Median Depth	Maximum Depth
Case 2, 130 Ft Over-burden							
Depth, Ft	20	55	110		20	55	110
Percent of Total	11	30	30	29	15	43	42
Average Swing Angle, Degree	135	50	100	180	120	45	90
Spot Bucket, Sec	3.0	3.0	3.0	3.0	3.0	3.0	3.0
Fill Bucket, Sec	10.0	10.2	10.2	10.2	9.5	9.5	9.5
Hoist Bucket, Sec	--	--	24.4	--	--	--	--
Swing Over, Sec	25.1	14.2	--	30.8	23.9	14.1	20.1
Swing Back, Sec	24.2	13.4	19.8	29.8	22.9	13.2	19.1
Total Time, Sec	62.5	40.8	57.4	73.8	59.3	39.8	51.7
Weighted Average Cycle Time, Sec		57.7				47.7	

operating time that could have been better utilized for useful cyclic operations. These are called operating delays (Table 43). Such delays can occur due to: 1.) machine movement, 2.) cleaning coal, and 3.) miscellaneous minor causes. An estimate of these delays for selected conditions is necessary to arrive at actual production time. The actual cycle time in practice is a weighted average of the chopping, minimum, maximum, and optimum-depth cycle times (Table 42).

SURFACE MINE DESIGN

The topography of the area determines the stripping method to be employed. For example, mountainous areas call for mountaintop removal or other contour mining methods, while flat terrain may dictate simple overcasting by area techniques. Seam thickness, as well as multiple-seam occurrences, must also be considered in equipment and method selection. Seam pitch, characteristics of the coal, the amount of groundwater over the coal seam, and stability of the highwall are also important factors. Finally, information concerning faults within the deposit, limits of the deposit, property lines, roads, surrounding private properties, and local land-use regulations must be carefully evaluated before actual mine design.

The economic life of the mine is related to pit planning by the yearly average stripping ratio and other factors that could assure profitability of the venture. Early return on the investment is desired and often pit designs are planned with this in mind. Equipment selection has direct financial consequences on pit planning. It may be possible to achieve low operating and maintenance costs with high initial costs. On the other hand, low initial capital investment will call for expansion capital in the future.

Pit entrances should be centrally located so that haulage distance between the center of gravity of the pit area and the dump is near minimum. Dead heading can be avoided by construction of several pit entrances preferably perpendicular to the cut.

Environmental concerns about surface mining have been on the increase in the last decade. Mine planning should include plans for reclamation procedures prior to the commencement of mining. Regulations require that reclamation be accompanied by revegetation and burial of toxic material to avoid groundwater contamination. Acid mine drainage must be treated and plans for its treatment initiated prior to mining. To gain public and community acceptance, special consideration should be given to the immediate community in hiring unskilled labor. Job training plans should include preparing those hired for more skilled jobs and positions of responsibility.

ENGINEERING DESIGN AND PIT LAYOUT

Generally, equipment is selected to conform with an existing mine design or mines are designed based on the anticipated equipment to be employed. In either case, the equipment performance characteristics and operating conditions determine the actual production levels. The following

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TABLE 43. WALKING DRAGLINE - OPERATING DELAYS (LEARMONT, 1975)

	2570-W 335-Ft Boom, 38° 300' Radius, 110 Cu Yd	2570-W 360-Ft Boom, 31-1/2° 340' Radius, 130 Cu Yd
Case 1, 110 Ft Overburden		
Approximate rate of face advance per hr, ft	10.8	8.8
Movement across face for a 30 ft advance, ft	80	80
Total dragline movement per hr. ft.	39.6	32.3
Move time per hour (1 min per 8 ft step including preparation), min	5.0	4.0
Clean coal time per hr, min	3.0	2.4
Miscellaneous delays per hr, min	4.0	4.0
Total operating delays per hr, min	12.0	10.4
Operating delay factor, %	80	82
Case 2, 130 Ft Overburden		
Approximate rate of face advance per hr, ft	6.6	7.4
Movement across face for a 30 ft advance, ft	160	80
Total dragline movement per hr, ft	41.8	27.1
Move time per hour (1 min per 8 ft step including preparation), min	5.2	3.4
Clean coal time per hr, min	1.8	2.1
Miscellaneous delays per hr, min	4.0	4.0
Total operating delays per hr, min	11.0	9.5
Operating delay factor, %	82	84

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is a set of definitions for use with the mathematical equations to be developed in this section. Since the use of English units is most common, the derivations here are in English units.

P = Production per month in tons
A = Reject fraction in run-off mine coal
U = Recovery fraction from coal in ground
COAW = Equivalent lb per cu yd of coal in the solid seam
(Usually = 2000 lbs)
TT = Equivalent lb per ton of coal
1 ton = 2000 lb for short ton
1 ton = 2240 lb for long ton
1 ton = 1000 kg for metric ton
T = Thickness of coal seam in ft
H = Overburden height in ft
H1 = Depth of dragline working bench in ft
SMH = Scheduled monthly operating hours
CT = Cycle time in seconds
Bs = Bucket size for shovel in cu yd
Bd = Bucket size for dragline in cu yd
BF = Bucket fill factor (fraction)
OF = Operating efficiency (fraction)
W = Pit width in ft
SP = Swell of material (percentage usually 25% to 30%)
Φ = Angle of highwall in degrees
Θ = Angle of spoil in degrees
FACD = MUF factor for dragline in lb/MUF
FACS = MUF factor for shovel in lb/MUF
Rs = Shovel reach in ft
Rd = Dragline reach in ft
Rsd = Shovel dumping radius in ft
Rdd = Dragline dumping radius in ft
Bb = Bucket size of bulldozer in cu yd
B1 = Bucket size of loader in cu yd
Bc = Bucket size of scraper in cu yd
S = Speed in feet per minute (fpm)
S1 = Loaded speed in mph
S2 = Empty speed in mph
LF = Load factor (fraction)
MF = Material factor (fraction)
MAT = Maintenance factor (fraction)
CF = Carry factor (fraction)
MPH = Available minutes per hour (fraction)
BCYPH = Maximum production per hour in cu yd
HD = Haul distance in ft
TCT = Total cycle time in sec
VL = Loose volume in cu yd
VB = Bank volume in cu yd
HPS = Hours per shift
SPD = Shifts per day
DPW = Days per week
WPM = Weeks per month

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Overburden Yardage to be Moved

Knowledge of the amount of overburden to be moved is the primary information on which the selection of a shovel or dragline is based. For a clean coal production of P tons per month, the acreage of coal to be exposed and the amount of overburden to be removed, calculate as follows:

P = Production per month in tons
A = Reject fraction in run-off-mine coal
U = Recovery fraction from coal in ground
COAW = Equivalent lb per cu yd of coal in the solid
T = Thickness of coal seam in ft
H = Overburden height in ft
ACP = Amount of run-of-mine coal to be mined in tons
ACE = Amount of coal to be exposed in tons
CYE = Cubic yards of coal to be exposed
SYE = Square yards of coal to be exposed
ACYE = The acreage of coal to be exposed
COB = Cubic yards of overburden to be removed

6-1/2"

$$\mathrm{ACP} = \frac{P}{1 - A}$$

9-1/8"

$$\mathrm{ACE} = \frac{\mathrm{ACP}}{U}$$

$$\mathrm{CYE} = \frac{(\mathrm{ACE})(\mathrm{TT})}{\mathrm{COAW}}$$

$$\mathrm{SYE} = \frac{(\mathrm{CYE})(3)}{T}$$

$$\mathrm{ACYE} = \frac{\mathrm{SYE}}{4840}$$

$$\mathrm{COB} = \frac{(\mathrm{SYE})(H)}{3}$$

Calculation of Bucket Size

Knowing the amount of overburden to be removed each month, a suitable bucket size can be calculated.

SMH = Scheduled monthly hours
CT = Cycle time in sec
OF = Operating efficiency
BF = Bucket-fill factor
NTCPM = Number of theoretical cycles per month

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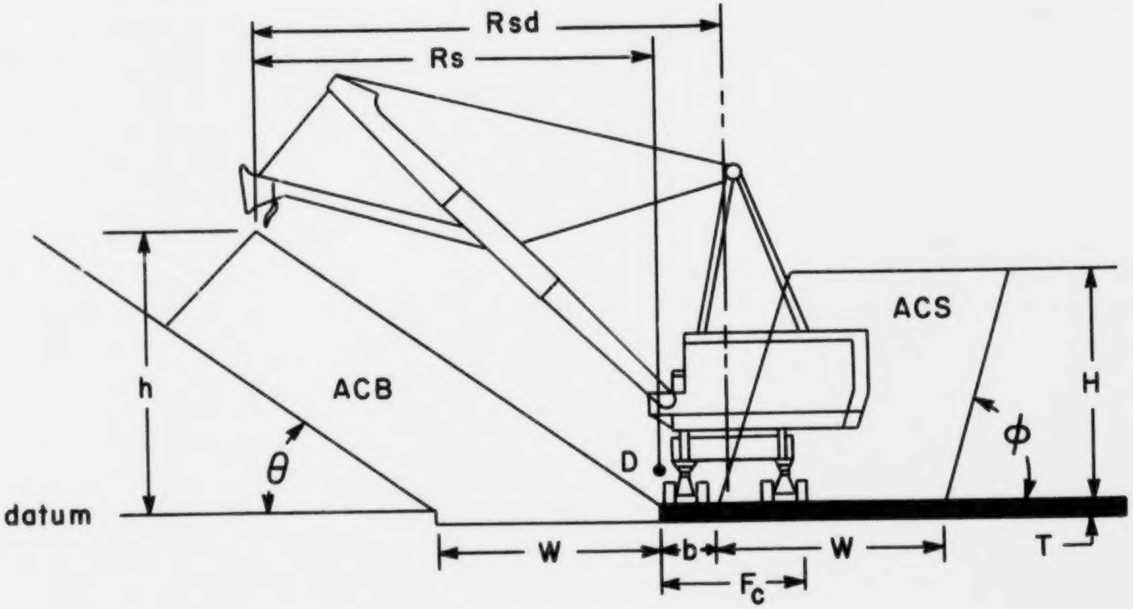

Figure 92. Shovel pit with coal fender (Stefanko, et al., 1973).

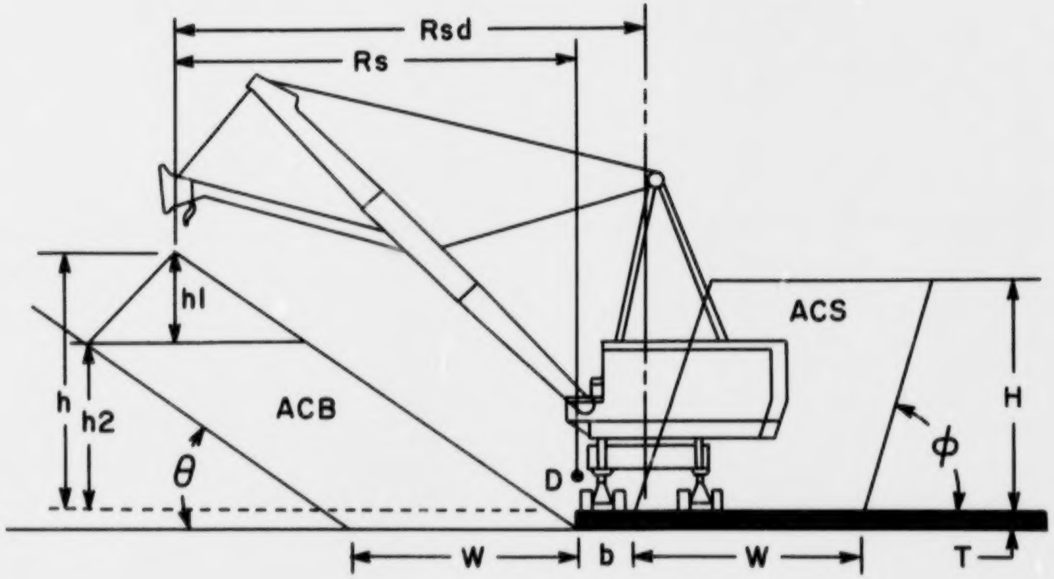
Rsd
Rs
ACS
ACB
hl
h
h2
θ
D
H
φ
T
W
b
W

Figure 93. Shovel pit without coal fender (Stefanko, et al., 1973).

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TBC = Theoretical bucket capacity
NBSIZE = The actual bucket capacity after considering the swell percentage, the bucket-fill factor and the operating efficiency.

$$NTCPM = \frac{(SMH)(3600)}{CT}$$

$$TBC = \frac{COB}{NTCPM}$$

$$NBSIZE = \frac{TBC}{(OF)(BF)}$$

It is now a common practice among operators to restrict the bucket sizes within the range of 70 to 140 cubic yards. Therefore, if the calculations should lead to a large bucket size, it is advisable to use two or more machines in multiple pits to meet the production requirements.

Calculation of Reach

Since the primary equipment can be a shovel or a dragline, the formulas applicable to both machines are developed here.

1. Shovel Reach--

The usual operating practice with a shovel is to align one set of its track on the rib of coal adjacent to the top of the spoil. An imaginary plane passing through this location would contain the point "D" shown above the front crawler in Figure 92. Projection of a horizontal line from the imaginary vertical plane to the point of discharge of the shovel dipper establishes the reach of the shovel. The reach is calculated as follows (Rumfelt, 1961). Attention is drawn to Figure 93.

ACS = Area cut in solid, sq ft
ACB = Area cut in broken (or spoil), sq ft
Φ = Angle of highwall, degrees
Θ = Angle of spoil, degrees
h = Height of spoil, ft
H = Height of overburden highwall, ft
T = Thickness of coal, ft
W = Width of pit, ft
b = Berm width, ft. This is assumed to be small and therefore is not included in the calculations.
F_c = Width across both crawlers in ft
Rs = Reach of shovel in ft
Rsd = Dumping radius of shovel in ft

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With reference to Figure 93

$$h_1 = \left(\frac{W}{2}\right) \text{Tan } \Theta$$

$$h_2 = h - \left(\frac{W}{2}\right) \text{Tan } \Theta$$

$$\text{ACB} = W \left[h - \left(\frac{W}{2}\right) \text{Tan } \Theta\right] + \left(\frac{W}{2}\right)\left(\frac{W}{2}\right) \text{Tan } \Theta + (W)\ (T)$$

$$= (W)(h) - \left(\frac{W^2}{2}\right) \text{Tan } \Theta + \left(\frac{W^2}{4}\right) \text{Tan } \Theta + (W)(T)$$

$$+ (W)(h) - \left(\frac{W^2}{4}\right) \text{Tan } \Theta + (W)(T) \qquad (1)$$

The corresponding area in solid is given by,

$$\text{ACS} = (W)(H) \qquad (2)$$

Assuming that the length is the same in the spoil as in the solid, it can be deduced that

$$\text{ACB} = (W)(H)\left(1 + \frac{SP}{100}\right) \qquad (3)$$

Therefore, equations (1) and (3) are the same, i.e.,

$$(W)(T) + (W)(h) - \left(\frac{W^2}{4}\right) \text{Tan } \Theta = (W)(H)\left(1 + \frac{SP}{100}\right)$$

and,

$$h = H\left(1 + \frac{SP}{100}\right) + \left(\frac{W}{4}\right) \text{Tan } \Theta - (T)$$

Hence, from the geometry in Figure 93

$$Rs = h \text{ COT } \Theta$$

which yields

$$Rs = \frac{1}{\text{Tan } \Theta} \left[H\left(1 + \frac{SP}{100}\right)\right] + \left(\frac{W}{4}\right) \text{Tan } \Theta - (T)$$

Defining F_c = the width across both crawlers in feet, the dumping radius of the shovel is given by,

$$Rsd = Rs + .5F_c$$

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2. Dragline Reach--

All definitions used in calculating shovel reach apply to the dragline case also. While the shovel operates on the top of the coal, the draglines work on a bench on the highwall or on top of the highwall itself. An imaginary vertical plane passing through the edge of the highwall would contain the point "D" shown to the left of the crawler in Figure 94. Projection of a horizontal line from the imaginary vertical plane to the point of discharge of the bucket content establishes the reach of the dragline. The reach is calculated as in the case of the shovel, with the following equations

Rd = Reach of dragline in ft
Rdd = Dumping radius of dragline in ft

$$h = H(1 + \frac{SP}{100}) + (\frac{W}{4}) \text{ Tan } \Theta - (T)$$

But the dragline reach as can be seen from Figure 93 is given by,

$$Rd = H \text{ COT } \Phi + h \text{ COT } \Theta$$

$$= \frac{H}{\text{Tan } \Phi} + \frac{1}{\text{Tan } \Theta} [H(1 + \frac{SP}{100}) + (\frac{W}{4}) \text{ Tan } \Theta - (T)]$$

Defining E_t = the outside diameter of the dragline tub, the dumping radius can be calculated as,

$$Rdd = Rd + .75E_t$$

3. Dragline with a Working Bench--

This case is illustrated in Figure 95. Here the dragline works on a bench at a depth H1 ft below the top of the highwall. This procedure may be practiced to increase the operating range of the dragline. The overburden to depth H1 is removed by some other equipment, and the reach calculation with consideration to bench is modified as follows:

$$Rd = \frac{H - H1}{\text{Tan } \Phi} + \frac{1}{\text{Tan } \Theta} [(H - H1)(1 + \frac{SP}{100}) + \frac{W}{4} \text{ Tan } \Theta - T]$$

$$Rdd = Rd + .75E_t$$

However, if the dragline is used for cutting this bench also, the dragline bucket size should be larger for the same clean coal production.

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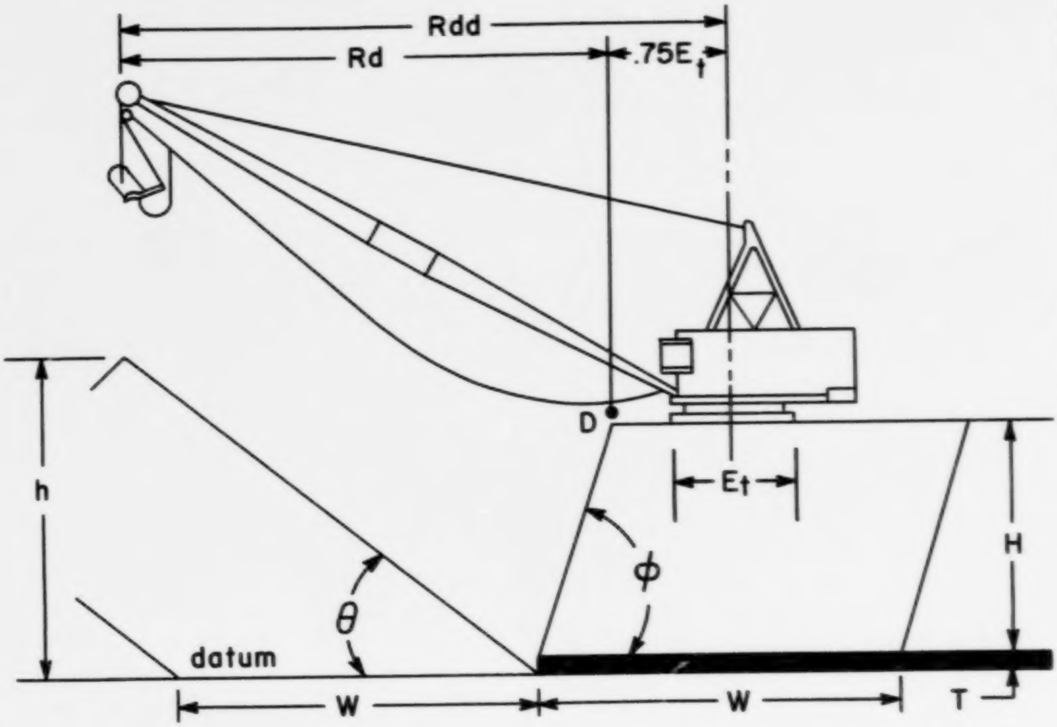
Rdd
Rd
.75E$_t$
D
h
θ
datum
W
W
T
H
ϕ
E$_t$

Figure 94. Dragline pit (Stefanko, et al., 1973).

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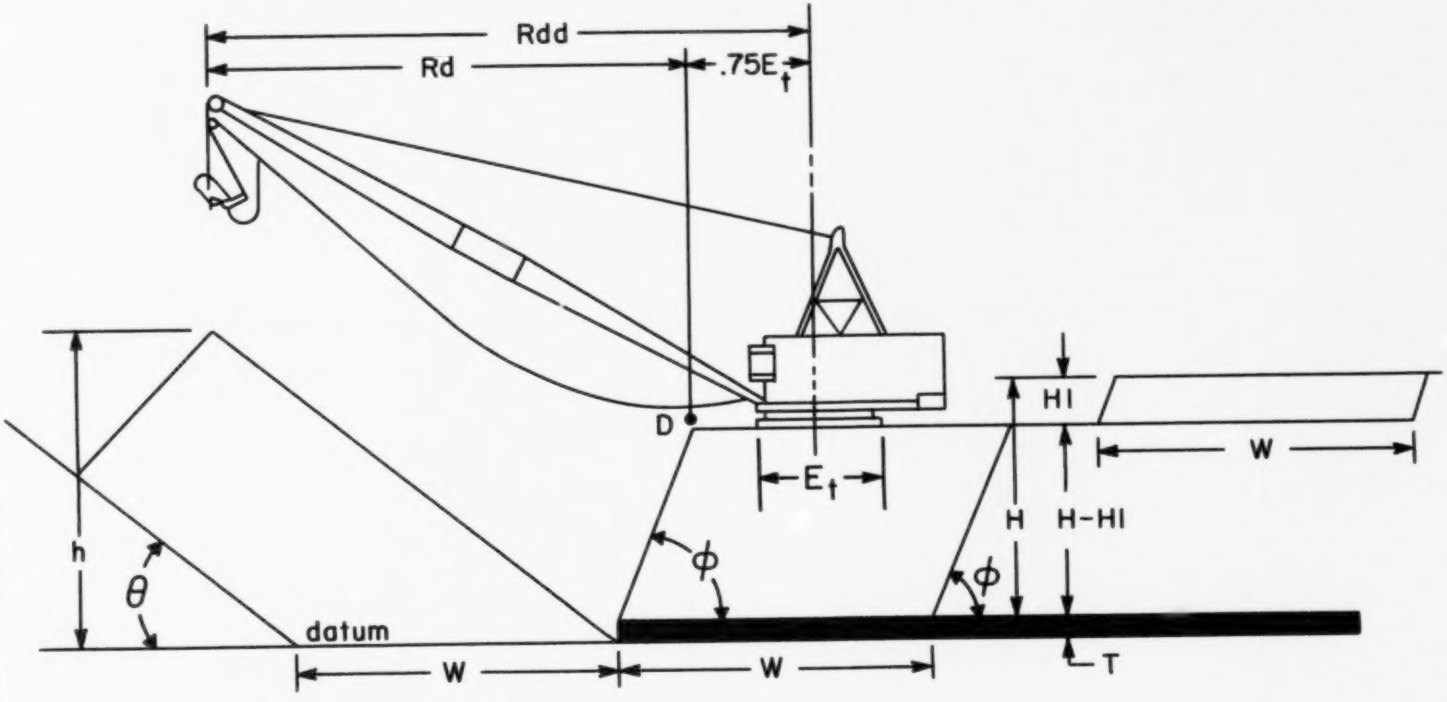
Rdd
Rd
.75E$_t$
D
HI
W
E$_t$
H
H-HI
ϕ
ϕ
h
θ
datum
W
W
T

Figure 95. Dragline pit with working bench (Stefanko, et al., 1973).

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Machine Weight Using the Maximum Usefulness Factor (MUF)

Maximum usefulness factor (MUF) is the measure of the capability of a machine to do a job. For a stripping machine, it is defined as follows:

MUF = Bucket size (cu yd) x Reach (ft)

Rumfelt (1961) investigated a large number of stripping machines and produced MUF curves for shovels and draglines. The curves of MUF plotted against machine weights are straight lines for different sizes of shovels and draglines, which implies that the weight of the machine in pounds divided by its MUF was a constant. The value for a constant was found to be approximately 745 for shovels (Figure 96). For draglines, the values varied between 467 and 575 (Figure 97).

Rs = Reach of shovel in ft
Bs = Bucket size of shovel in cu yd
FACS = MUF constant for shovel in lb/MUF
SMW = Weight of shovel in lb
SMW = (Rs)(Bs)(FACS)
Rd = Reach of dragline in ft
Bd = Bucket size of dragline in cu yd
FACD = MUF constant for dragline in lb/MUF
DMW = Weight of dragline in lb
DMW = (Rd)(Bd)(FACD)

Dragline Digging Its Own Bench

The dragline sometimes digs its own bench in order to increase its range. In this type of operation, the machine chops material above its operating bench and therefore, the cycle time is no longer the normal cycle time but rather a weighted average cycle time. This can be calculated from the normal and the bench "chopping" cycle times as follows:

SMH = Scheduled monthly hours
NTCPM = Number of theoretical cycles per month
COB = Cubic yards of overburden to be removed
TBC = Theoretical bucket capacity in cu yd
NBSIZE = The actual bucket capacity in cu yd
OF = Operating Efficiency
BF = Bucket fill factor
CT = Normal cycle time in sec
CTB = Cycle time for "chopping" its bench in sec
CTW = The weighted average cycle time in sec
H1 = Depth to dragline bench in ft
H = Overburden height up to dragline bench from coal in ft
Ht = Total height of the overburden in ft
HT = H + H1

$$CTW = CT\left(\frac{H}{H1}\right) + CTB\left(\frac{H1}{HT}\right)$$

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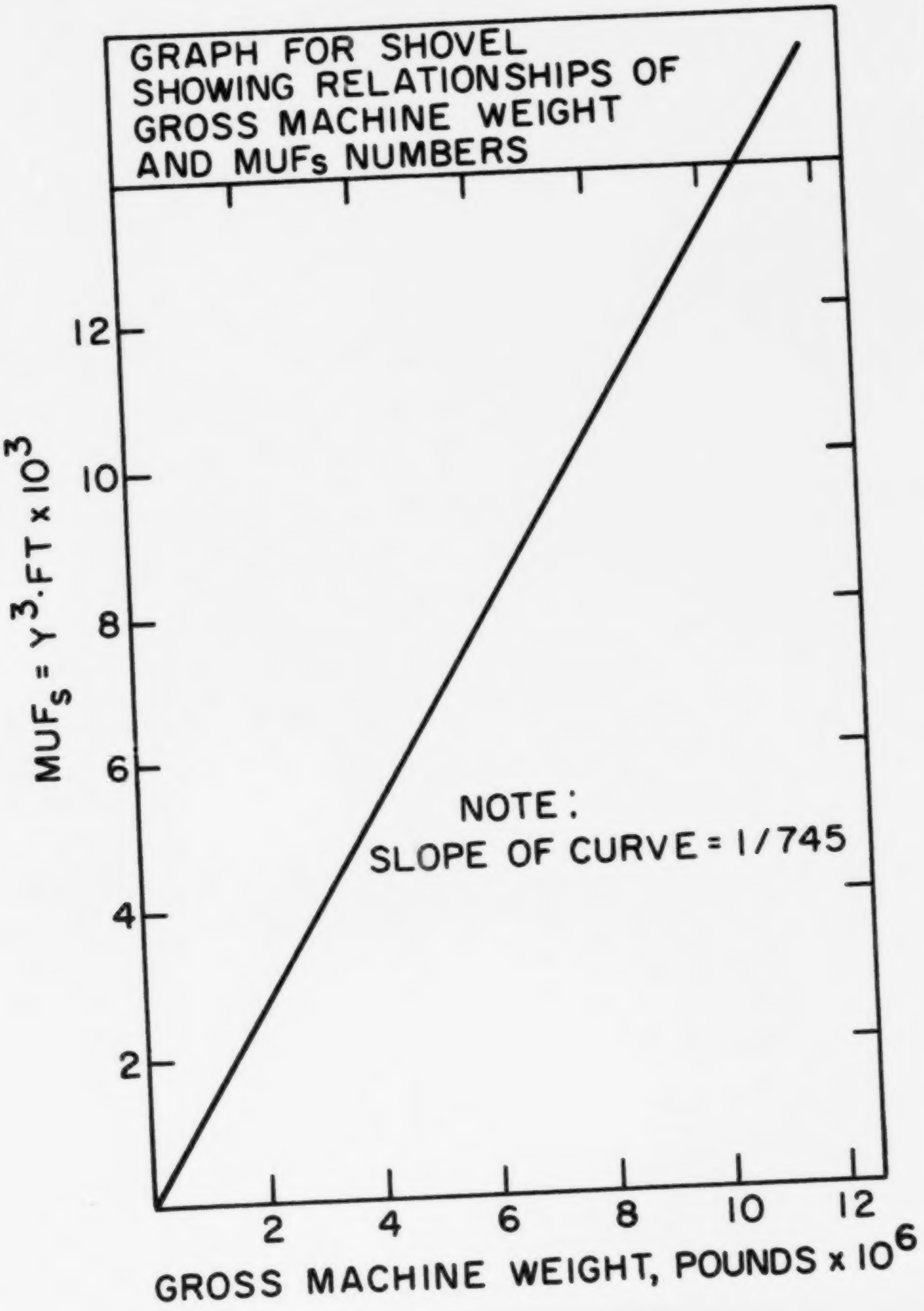

Figure 96. Shovel MUF graph (Rumfelt, 1961).

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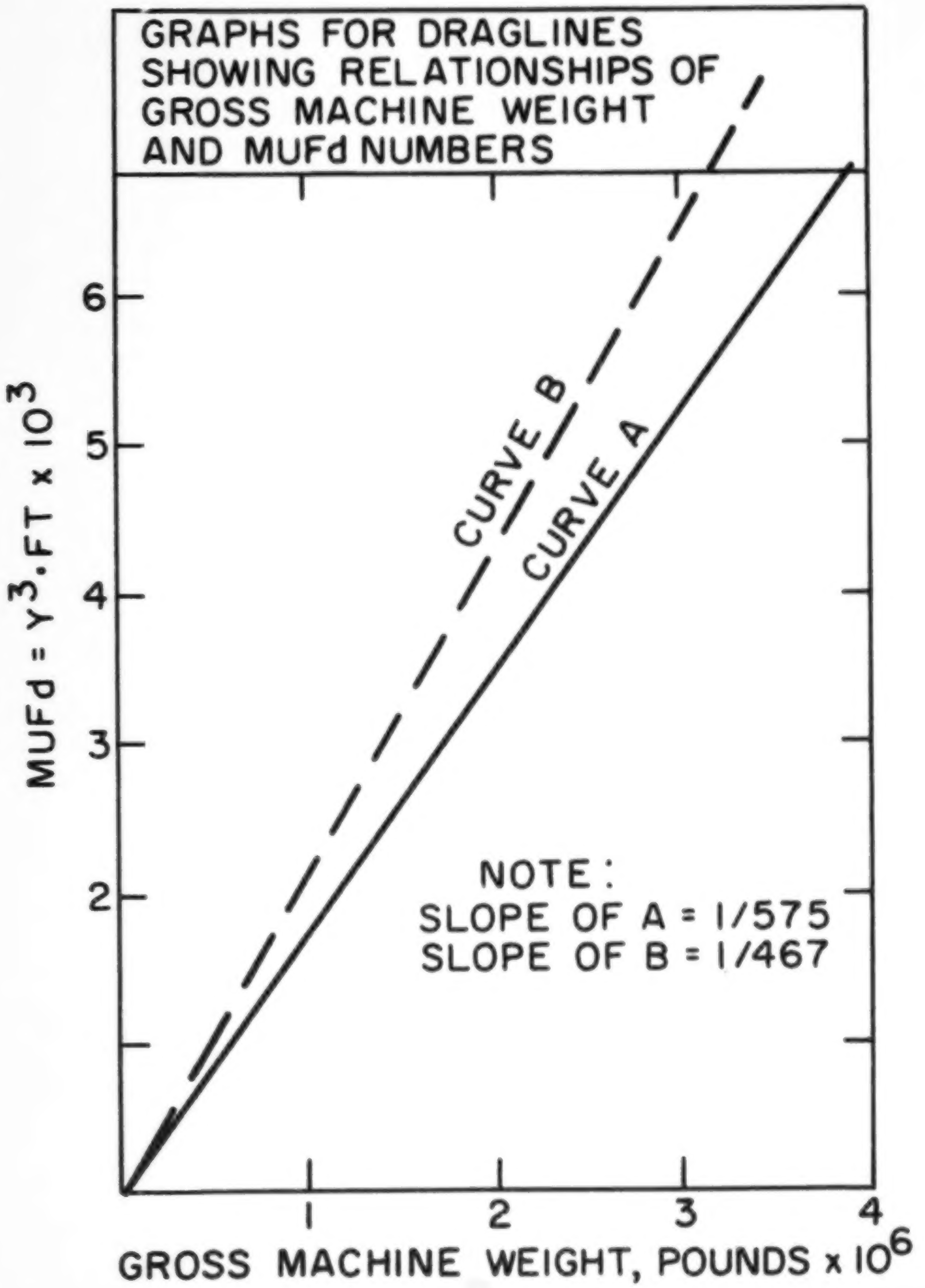

Figure 97. Dragline MUF graph (Rumfelt, 1961).

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$$\mathrm{NTCPM} = \frac{(\mathrm{SMH})(3600)}{\mathrm{CTW}}$$

$$\mathrm{COB} = \frac{(\mathrm{P})(\mathrm{TT})(\mathrm{HT})}{(1-\mathrm{A})(\mathrm{U})(\mathrm{COAW})(\mathrm{T})}$$

$$\mathrm{TBC} = \frac{\mathrm{COB}}{\mathrm{NTCPM}}$$

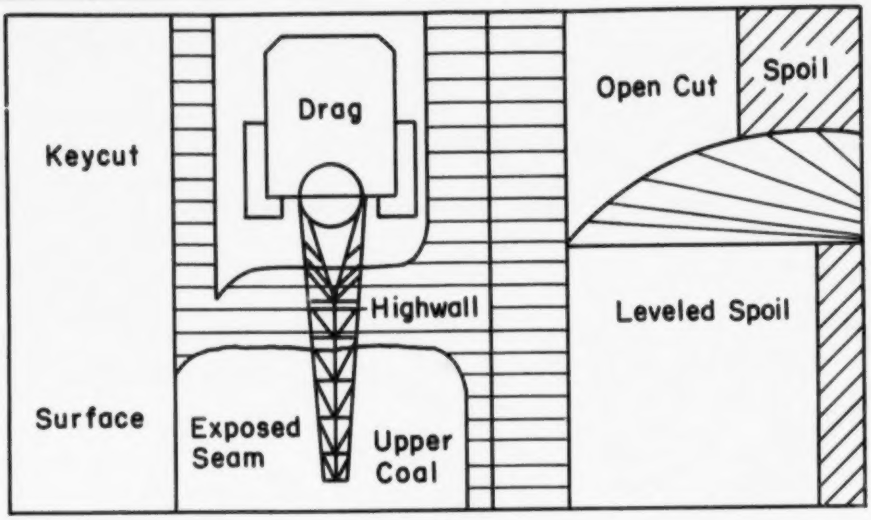
$$\mathrm{NBSIZE} = \frac{\mathrm{TBC}}{(\mathrm{OF})(\mathrm{BF})}$$

If the dragline digs its own bench in an operation, the weighted cycle time for both the bench digging and the normal operation will be greater than the normal cycle time. The net result in the selection of the dragline is one with a larger bucket size. The reach of the dragline must be calculated for the total overburden height. It is obvious that both of these will increase the maximum usefulness factor and therefore, the machine weight (Figure 95).

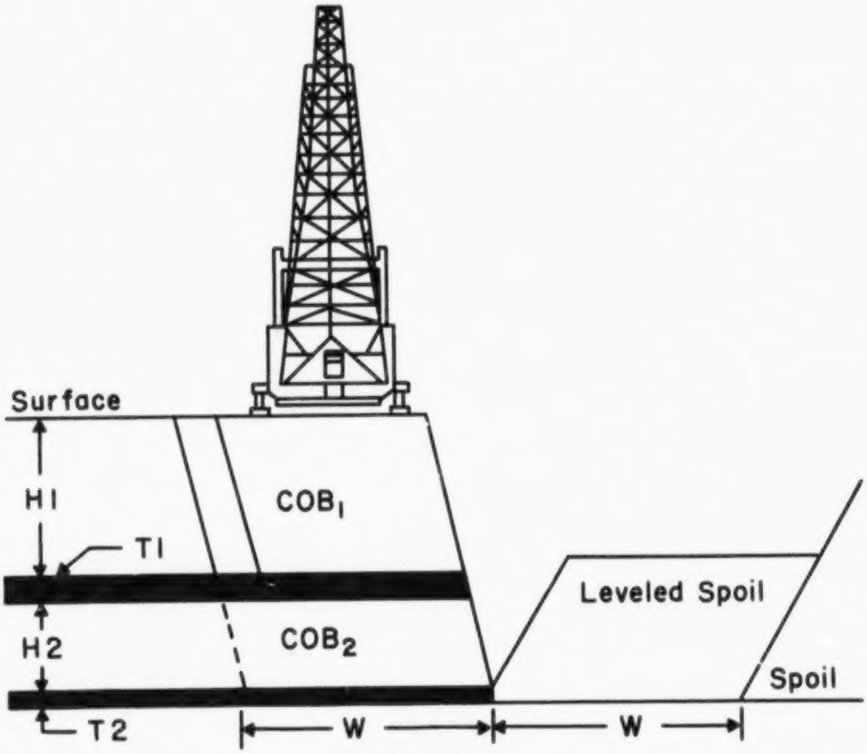
Multiple-Seam Mining

In multiple-seam mining, the dragline removes the overburden over the first coal seam, casts the spoil, and the coal seam is loaded out. Following this, the top of the spoil is leveled to form an operating bench for the dragline. The dragline is repositioned on this bench to strip the parting. The dragline action here is a chopping rather than a digging one. In chopping, the bucket fill factor decreases because of reduced contact angle between the bucket teeth and the overburden. Also, there is usually an increase in the bucket swing angle; i.e., when the dragline works from the leveled spoil it has to swing more than the normal swing angle in order to dump the parting material behind the original spoil. With reference to Figures 98 and 99, the weighted average cycle time can be calculated as follows:

SMH = Scheduled monthly hours
BF = Bucket fill factor
T = Coal thickness in ft
H = Height of overburden in ft
P = Production per month in tons
TT = Equivalent lb per ton of coal
COAW = Equivalent lb per cu yd of coal in solid
A = Reject fraction in run-off mine coal
U = Recovery fraction from coal in ground
COB = Cubic yards of overburden removed
CT = Normal cycle time of dragline in sec
CT2 = Cycle time for chopping overburden 2 in sec
CTM = The weighted cycle time in sec
P1 = Production from seam 1 in tons


Keycut
Drag
Open Cut
Spoil
Highwall
Leveled Spoil
Surface
Exposed Seam
Upper Coal

Plan View


Surface
H1
COB$_1$
T1
H2
COB$_2$
Leveled Spoil
Spoil
T2
W
W

Section View

Figure 98. Multiseam dragline operation with upper seam exposed (Stefanko, et al., 1973).

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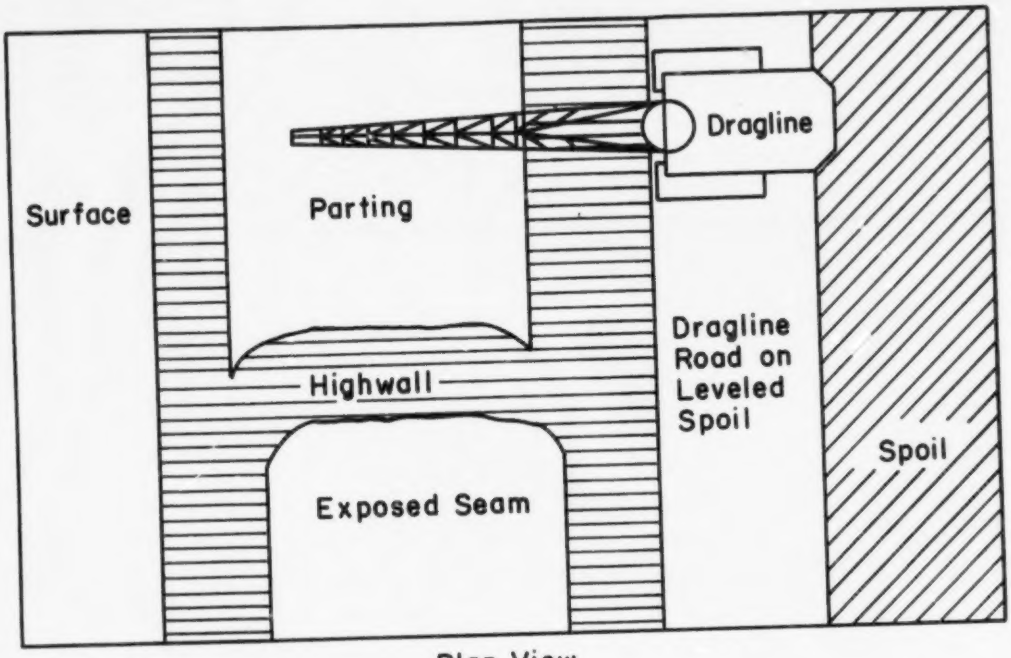
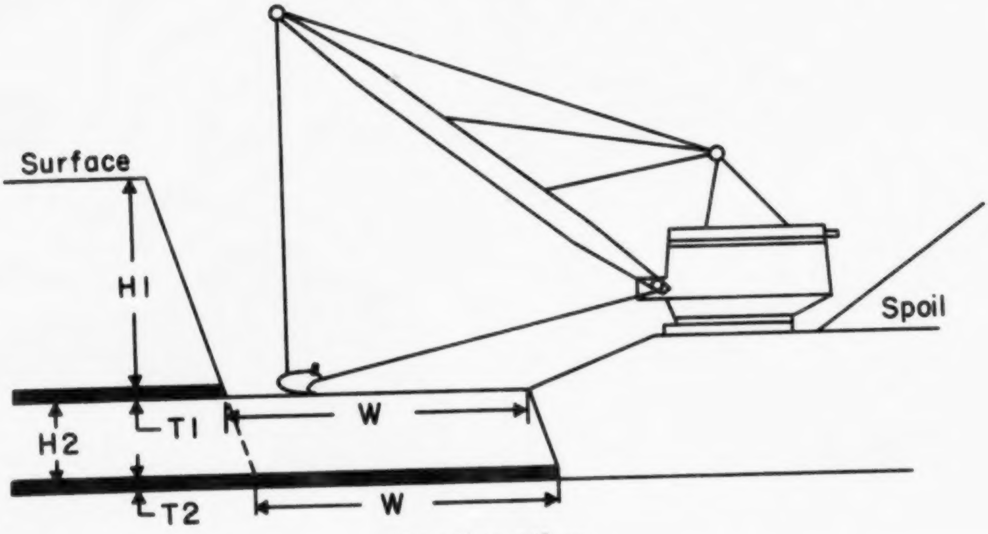

Dragline
Surface
Parting
Dragline Road on Leveled Spoil
Highwall
Spoil
Exposed Seam

Plan View


Surface
H1
Spoil
T1
W
H2
W
T2

Section View

Figure 99. Dragline positioned on leveled spoil removing parting (Stefanko, et al., 1973).

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P2 = Production from seam 2 in tons
H1 = Height of overburden 1 in ft
H2 = Height of overburden 2 in ft
T1 = Thickness of seam 1 in ft
T2 = Thickness of seam 2 in ft
BF1 = Bucket fill factor for seam 1
BF2 = Bucket fill factor for seam 2
COB1 = Cubic yards of overburden removed in 1
COB2 = Cubic yards of overburden removed in 2

$$T = T1 + T2$$

$$H = H1 + H2$$

$$P = P1 + P2$$

$$CTM = CT \left(\frac{H1}{H}\right) + CT2 \left(\frac{H2}{H}\right)$$

$$NTCPM = \frac{(SMH)(3600)}{CTM}$$

$$BF = BF1 \times \frac{H1}{H} + BF2 \times \frac{H2}{H}$$

$$P1 = P \left(\frac{T1}{T}\right)$$

$$P2 = P \left(\frac{T2}{T}\right)$$

$$COB1 = \frac{(P1)(TT)(H1)}{(1 - A)(U)(COAW)(T1)}$$

$$COB2 = \frac{(P2)(TT)(H2)}{(1 - A)(U)(COAW)(T2)}$$

$$COB = COB1 + COB2$$

$$TBC = \frac{COB}{NTCPM}$$

$$NBSIZE = \frac{TBC}{(OF)(BF)}$$

The total effect of multiple-seam mining on dragline selection is that the weighted cycle time increases, the number of passes per hour decreases, the bucket-fill factor decreases, and the bucket size increases. Therefore, the maximum usefulness factor increases. Even if the reach remains the

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same, the machine weight will increase to meet the required production per month. The calculation of the reach in this case is involved since the dumping positions for the two modes of operation are different.

BUCKET-WHEEL EXCAVATOR SELECTION

The design of a bucket-wheel excavator (BWE) for the excavation of ore or waste is determined by the digging power required and the abrasiveness of the material to be excavated. A BWE consists of too many assembly groups and has to fulfill too many simulataneous functions to allow for the calculation of any individual component by one definite standard formula. Lengths and dimensions of a wheel excavator are highly interrelated, and their combination is always a compromise between the designed performance of the machine and its cost. In general, these machines are job-tailored and, as for every other type of excavating equipment, the cutting, tearing, loosening and splitting force required to excavate material is the deciding factor in the design of a BWE. In the United States, wheels are operated in tandem with shovels and/or draglines thereby necessitating considerations of equipment matching and working area limitations in pit design. Since bucket-wheel excavators are essentially German development, the standard units associated with BWEs are in the metric system. Therefore, in the discussion here metric units are used.

SOIL CLASSIFICATION

The BWE is an ideal machine for working in soft and medium hard strata, and it is gradually being developed for use in harder strata. Specifically, the digging resistance of the soil is the most sensitive parameter affecting the BWE operation. For excavation purposes, soils can be classified as follows:

1. The soils that can be excavated by a spade, e.g., unconsolidated strata like sand, gravel and silt.
2. The soils that can be loosened by a pickaxe before excavation, e.g., partly consolidated strata like clay beds.
3. The soils that can be loosened by blasting before excavation, e.g., hard shales, sandstone, etc.

The above classification is broad and general and cannot be of specific assistance to the BWE designer. Classification of soils according to their diggability is vital; but unfortunately, there is not any clearcut, standard method for determining this factor. Some common methods of expressing the digging resistance are:

1. Kg/cm, where the cutting resistance is calculated on the basis of load per unit length of the cutter in contact with the soil.
2. Kg/cm^2, where the load is related to the area of the cut slice.

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3. Kg/cm^3, where the digging resistance is calculated as a function of the excavated volume per bucket.

The theories that are being advanced and the discussions taking place on what is considered to be the most suitable reference quantity vary widely. So far, most design calculations have been based on the cutter lengths or on the cross-sectional area of the slice cut, i.e., methods 1 and 2; but it must be recognized that the volume of material excavated per bucket has no small influence on the digging resistance. At the Fifth International Earthmoving Conference held in Prague in October, 1963, Professor N. G. Dombrowsky (1964) gave the following specific digging forces for various types of ground:

a. Light ground	1.8 to 2.5 kg/cm^2
b. Medium ground	3.0 to 3.5 kg/cm^2
c. Heavy ground	7.0 to 18. kg/cm^2

While it is not uncommon in non-German technical literature to consider the resistance to digging in relation to the cross-section of the slice, manufacturers of wheel excavators in Germany are still largely using the effective cutter length for determining the cutting forces. This is so because of the large number of test results available on various soils. Based on 150 large-scale tests of BWE operations, Himmel (1961) has calculated the specific cutting forces required to dig the types of ground encountered in open-cut mines. On an average, these were found to be:

1. About 20 kg/cm for light ground such as sand and gravel.
2. About 30 to 40 kg/cm for medium heavy ground such as sandy loam, pure loam, loess, lean clay, etc.
3. About 50 to 60 kg/cm for heavy ground such as heavy compact and plastic clays.

It is not possible to draw any conclusions on the above proposed values because the cutting power depends on the type of ground, the cross-sectional area and the shape of the slice cut, the configuration and sharpness of the cutter blades, the shape of the teeth, and the cutting speed. Hard ground requires a high specific cutting force, high cutting speeds, and additional cutting blades between the buckets. Reducing the output and the rate of swing decreases the slice cross-section and increases the specific cutting force for a given wheel drive rating. This will be clear from the following formula (Rasper, 1964):

$$N_G = \frac{K}{\eta \times c} \sqrt{Q \times S \times R}$$

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where,

N_G = Power required for cutting in kw
Q = Actual digging capacity in cubic meters per hour
S = Number of bucket discharges per minute
R = Radius of the cutting wheel in meters
c = Constant depending on the bench height/wheel diameter ratio. For the cutting height = 2/3 wheel diameter, its value is 171
η = Efficiency of the wheel drive
k = Specific cutting force in kg/cm

Shown in Tables 44 and 45 are reference values for specific cutting forces applied to the more important bulk materials and virgin soils. The tables must be viewed with reservation as to the nonhomogeneity of the different soils from one geographical location to the other.

Devising methods for measuring the cutting resistance is another difficulty because the results from such theoretical considerations vary with the size of the sample tested, the angle of loading, the rate of loading and other such experimental procedures. The methods commonly employed for the determination of the cutting resistance are:

1. Simulating equipment pulled through the soil to be excavated for direct measurement of the force applied.
2. Laboratory tests with tooth and tooth impression measurements on the rock or soil samples.
3. Determination of the compactness by Proctor needles.

In a method developed by a German manufacturer (LMG), the entry of an excavator tooth into soil is copied in the laboratory on a soil specimen (Rasper and Ritter, 1961). This gives some information with respect to the soil split resistance. If numerous tests have been made with soils on which the cutting effect of BWE has been studied and analyzed, a correlation can be drawn between the theoretical laboratory values and the actual digging resistance. From observations so far, theoretical analysis can at best serve only as a guideline in the selection of the wheel because of the vast complex of influences which exist in practice that cannot be considered theoretically. Practical tests on the site of employment will confirm the suitability of the machine selected for a particular job.

In tests conducted on glacial till, sandstone and devonian shale, it was confirmed that earth containing a high percentage of boulders is unsuitable for wheels (Cheek, 1966). The capacity of any excavator operating in earth material with boulders will be affected adversely in proportion to the frequency of boulder occurrence. Two possible reasons can be attributed to this poor performance. Some boulders, like basalt and granite, are too hard for economical cutting. On the other hand, boulders and rocks composed of materials which the machine can cut in the solid state become loosened during excavation and cannot be rehandled.

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TABLE 44. SPECIFIC CUTTING FORCES OF VIRGIN SOILS FOR BWE EXCAVATION (GARTNER, 1965)

SOIL TYPE	SPECIFIC CUTTING FORCES KG/CM
Earth	10 - 30
Loess	20 - 40
Sand (fine, coarse, wet or dry)	10 - 40
Clayey sand	10 - 50
Gravel fine	20 - 50
Gravel coarse	20 - 80
Sandy loam and wet loam	20 - 60
Dry loam	20 - 80
Clay wet	30 - 65
Clay dry	50 - 120
Clay schistose	35 - 120
Sandy clay	20 - 65
Clayey slate	50 - 160
Slate	70 - 100
Sandstone (easy digging)	70 - 160
Sandstone (hard digging)	160 - 280
Gypsum	50 - 130
Lime	30 - 120
Phosphate	80 - 200
Marl	60 - 140
Limestone	100 - 180
Weathered granite	50 - 100
Alluvial light consolidation	30 - 60
Alluvial heavy consolidation	70 - 150
Alluvial medium consolidation	50 - 80
Hard coal normal	50 - 100
Hard coal frozen	100 - 160
Lignite	20 - 70
Brown iron ore	190 - 210

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TABLE 45. SPECIFIC CUTTING FORCES OF MATERIALS SUITABLE FOR BWE LOADING OPERATIONS (GARTNER, 1965)

MATERIAL TYPE	SIZE mm	SPECIFIC CUTTING FORCE KG/CM
Sand		10 - 20
Gravel fine	0 - 100	20 - 25
Gravel coarse	100 - 400	10 - 25
Earth		10 - 15
Phosphate		10 - 20
Bauxite		10 - 20
Iron-Mn-Cr	0 - 150	20 - 40
Ores	150 - 300	20 - 50
	300 - 450	20 - 80
Loose hard coal		10 - 30
Solidified hard coal		10 - 40
Lignite		10 - 25
Coke		10 - 30
Pellets (ore; cement)		10 - 25
Limestone		10 - 30
Slag		10 - 25

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BWE OUTPUT CONSIDERATIONS

The theoretical output of a BWE is based on the bucket size and the number of bucket discharges per minute. If

I = Nominal bucket capacity in cubic meters
Z = Number of buckets in the wheel
V_1 = Cutting speed of the wheel in meters per second
D = Diameter of the wheel in meters,

then

$$S_s = V_1 \times Z/\pi D$$

where

S = Number of bucket discharges per second

$$Q_t = I \times S_s \times 3600$$

and

Q_t = Theoretical capacity of the excavator in cubic meters per hour.

As can be seen from the above equations, the number of bucket dischargers is dependent on the peripheral speed. The peripheral speed of a bucket wheel is limited by the ability of the wheel to discharge its bucket content on the chute against the counteracting centrifugal force. In theory, the maximum peripheral speed must be such that the bucket discharge will be ensured. Mathematically,

$$M \times g = M \times V_1^2/ R$$

where

M = Mass of material in the bucket, kg
R = Radius of the wheel in meters
g = Acceleration due to gravity, meters per second^2

which yields the following expression:

$$V_1 = \sqrt{g \times R} = V_{max}$$

Practical values of speed lie between 0.4 V_{max} to 0.6 V_{max}. To keep the wear on bucket's cutting knives or teeth at a minimum, speeds do not exceed 5 meters per second. The peripheral speed selected will also depend greatly on the nature of material to be excavated. In principle, however, a higher peripheral speed will be decided upon if hard material is to be

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cut, in which case maximum output may not be attained. Based on a constant output, the doubling of the peripheral speed will halve the amount of material excavated by each bucket; thus, cutting performance will be reduced.

Yet another factor that affects the output of a BWE is the bucket-filling capacity. It has been found that, in hard ground, the bucket-filling is around 30% to 40% of the nominal capacity. The relationship between digging resistance and the hourly capacity of the BWE is

$$Q_1/Q_2 = k_2^2 / k_1^2$$

where

Q_1 = BWE hourly capacity in cubic meters in soil with specific cutting resistance k_1
Q_2 = BWE hourly capacity in cubic meters in soil with specific cutting resistance k_2

Thus, the actual capacity of the BWE in any soil is given by:

$$Q_a = I \times B_f \times S_s \times 3600$$

where

B_f = Bucket-filling capacity in the soil expressed as a fraction of the nominal bucket capacity
S_s = Number of bucket discharges per second
Q_a = Actual capacity of the BWE in cubic meters/hour

In soils with high cutting resistance, higher cutting speeds with lower bucket-filling will result in a very small Q_a as compared to Q_t. The ratio may be as small as 0.2. One can visualize that in hard soils the excavating operation of the BWE has changed to a milling operation.

In most BWE calculations, the ratio of the bench height to the wheel excavator diameter is taken at two-thirds because the machine performance is optimum at this ratio. Obviously, with a lesser bench height full advantage of the wheel capacity is not taken. Also, increases in bench height above this limit result in undercutting which may lead to excessive spillage to be rehandled; and in soft soils sliding of the burden over the whell may result. The specific cutting resistance of the soil which can be calculated for a given digging power rating from the following formula,

$$K = \frac{N_G \times \eta \times C}{\sqrt{Q \times S \times R}}$$

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where

K = Specific cutting resistance in kg/cm
S = Number of bucket discharges per minute
Q = Actual capacity in the soil in cubic meters/hour
R = Radius of the bucket wheel in meters
N_G = Digging power rating of the drive in kw
η = Efficiency of the drive
C = Constant of the bench height to wheel diameter ratio

is found to be increasing with decreasing bench height for a given wheel diameter. The change in the value of "C" with a change in the height/diameter ratio can be observed in the following table (Rasper, 1964).

Height/Diameter Ratio	0.1	0.2	0.3	0.4	0.5	0.6	0.67	0.7
Value of "C"	295	248	222	203	189	178	171	168

BUCKET-WHEEL DESIGN

At present, two types of bucket design are recognized, cellular and noncellular. In noncellular wheels, a ring chute isolates the bucket from the material flow behind it during the digging and lifting motion. Material is carried in the bucket to a point where the ring chute clears the way for the flow of material to the ladder conveyor. In the cell-type wheels, a buffer or cell is arranged behind each bucket for the excavated material. The cells side discharge this material onto a ladder conveyor. A cell-type wheel is usually stronger and more resistant to wear; but with the application of BWEs to harder formations which require higher cutting speeds, the cell-less wheels have superseded the cellular wheels. This preference for cell-less wheels arises because the discharge of excavated material from cells is governed by the rotary motion of the wheel. At high speeds, cellular wheels do not discharge completely.

The most important factor in bucket-wheel design is the diameter of the wheel. Much stress is placed on the correct size of the wheel since the service weight of the machine varies as the square of the diameter; cutting torque varies inversely as the diameter; and the cutting speed should never exceed $\sqrt{0.2gD}$, where g is the acceleration due to gravity and D is the wheel diameter. The diameter should also be sufficient to accommodate the chute and roller bar and to transfer material from the buckets to the ladder-conveyor. Since the ladder-conveyor is taken beyond the axle of the wheel, it is a controlling factor in wheel design.

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OTHER CONSIDERATIONS

Wheel-boom and discharge-boom lengths have to be derived from operating conditions. The length of the wheel boom depends on the specified cutting height above and cutting depth below the grade and on the block width. The gradient limitation on the ladder-conveyor is an important factor which is decided by the type of material being handled. The length of the discharge boom depends on the amount of conveying necessary and the height, or possibly, the depth of discharge below grade. In an open-cut mine, the length of the discharge boom should be such that "spoiling" is properly affected on the dump zone and not within the pit limits.

Restrictions on the size of a modern-day BWE are imposed not by the design of the wheel but by the conveying of material from the bucket to the discharge point on the stacker boom. Also, with the design of BWEs for deep mines, a dependable steep angle conveyor is necessary. Belt speeds of 300 to 400 meters per minute and belt widths of 200 centimeters are not uncommon. In cases where the inclination angle of the ladder boom is more than 35 degrees, sandwich belt conveyors have proved to be highly satisfactory. Because of the types of material that the BWE handles, the conveyors must be liberally dimensioned and properly maintained.

MODE OF OPERATIONS

Modern BWEs generally excavate in blocks. Figure 100 shows a BWE working in an established cut. The wheel is positioned to travel on the pit floor in line with the top edge of the old highwall. As it advances, a new highwall is exposed in the direction of excavation. Digging is done by rotating the wheel, swinging it from side to side in long parallel arcs, and "crowding" into the bank, by advancing the entire machine, or by the travel of the digging boom if an automatic crowd is available (Figure 100). A second way by which the wheel can be advanced into the bank is by the falling cut method. A brief description of each of these methods is given below.

Cut with a Crowding Machine

At the end of every swing, the digging boom can be extended by the thickness of cut desired and the boom swung back in the reverse direction. Obviously, the thickness of bank excavated does not vary with the boom position; therefore, the slewing motion of the boom is fairly constant for uniform output. The thickness through which the digging boom can be advanced into the bank is theoretically calculated from the formula (Mani, 1966):

$$t = 0.133 \times \sqrt{Q_a / R \times S}$$

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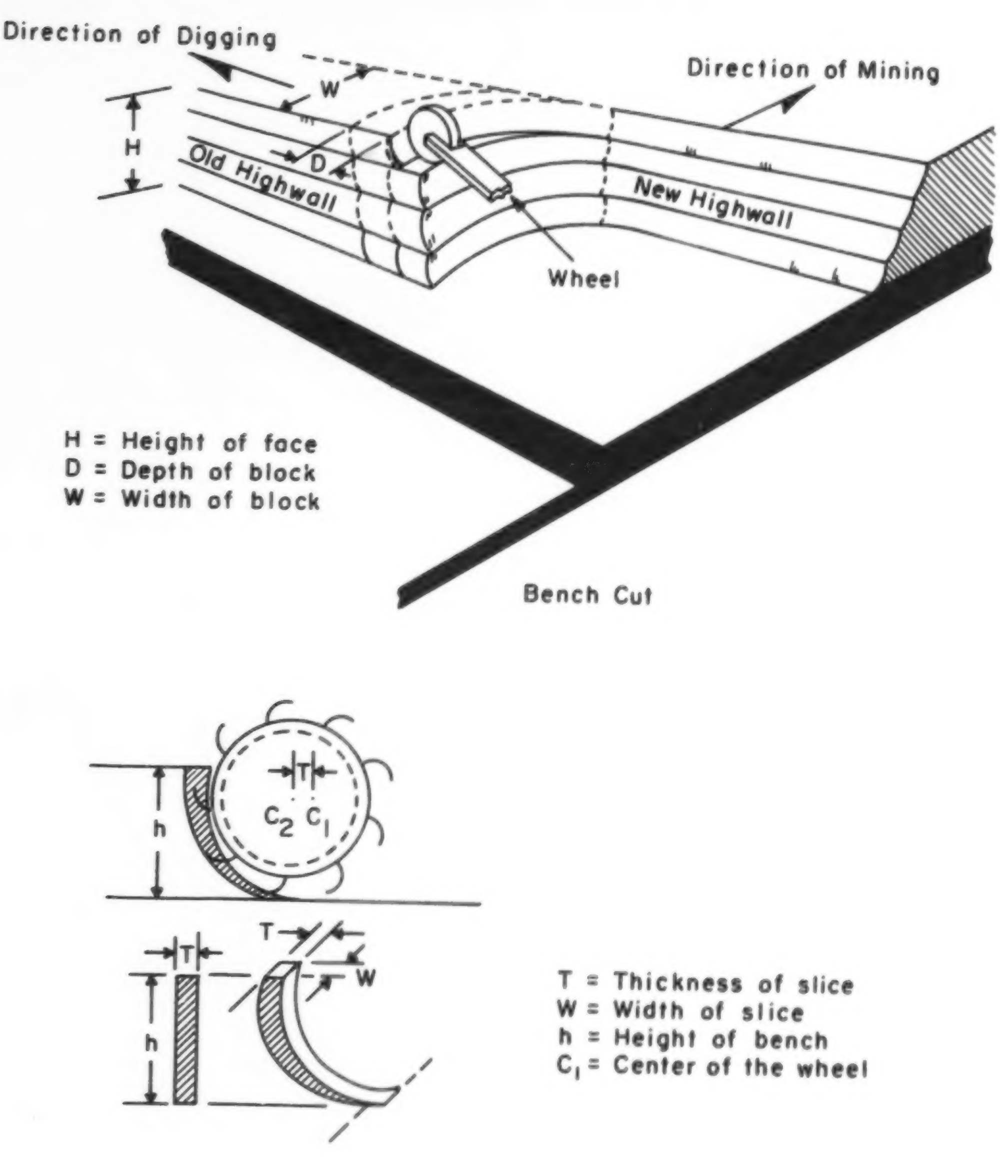
Direction of Digging
W
Direction of Mining
H
D
Old Highwall
New Highwall
Wheel
H = Height of face
D = Depth of block
W = Width of block
Bench Cut
T
C2 C1
h
T
T
W
h
T = Thickness of slice
W = Width of slice
h = Height of bench
C1 = Center of the wheel

Figure 100. Block digging operation with BWE.

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where

Q_a = Actual capacity of the excavator in cubic meters/hour
S = Number of bucket discharges per minute
R = Radius of the wheel in meters
t = Thickness of slice in meters

Cut with a Noncrowding Machine

The essential difference between this cut and the former is that here the machine is moved forward bodily in the direction of the bench by the thickness of the cut. The slew axis progresses by successive distances, t, along the line of advance; and the successive segments that are cut are not parallel but sickle-shaped. The thickness of slice varies constantly with angle Θ, both to the right and to the left of the direction of advance. A very close approximation of slice thickness at any point P is given by:

$$t_p = t\ (\mathrm{Cos}(\Theta) + \frac{\mathrm{Sin}^2\Theta}{2P_1}\ )$$

where

$P_1 = S_r/t$
S_r = Slewing radius in meters
t = Thickness of slice in meters
Θ = Angle between the direction of advance and direction of P

Therefore, to maintain a uniform output as the boom slews away from the center, the loss in the thickness of slice must be compensated for by increasing the width of the slice cut. This is achieved by increasing the slewing speed such that $V_p = V/\cos(\Theta)$, where V_p is the slewing speed at the point P, and V is the starting slewing speed in the direction of the advance.

Two factors limit the angle of slewing. At angles more than 70 degrees, the slewing speed increases disproportionately to the increase in output; therefore, usually with noncrowding machines, the width of the block is such that the swing on either side of the direction of advance is maintained within limits. The width of the bench is given by:

$$S_r(\mathrm{Sin}\ \Theta_L + \sin\ \Theta_R)$$

where

S_r = Radius of slewing in meters
Θ_R = Angle to the right of the direction of advance $\leq 70°$
Θ_L = Angle to the left of the direction of advance $\leq 70°$

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Falling Cut Method

Depending on stratification and fissuring, it is often advantageous in firm ground to operate by the drop or falling cut method. At the commencement of a block of maximum face height, a top cut about half the wheel diameter is taken by the bench cut method, the depth of the cut being great enough to leave a face formed for the top cut of the subsequent block. After this initial preparation, the excavator retreats a short distance and the rest of the block is excavated in segments which are proportioned by lowering the wheel to the desired thickness (t). The slice taken here is very similar to that of the bench cut. The geometry of the cut is explained in Figure 101 including the slice cross-section at the line of advance. If

A_h = Depth of slice normal to the working face at 0° slew in meters
A = Depth of advance in meters

the variation in the sickle depth in the falling cut method can be expressed as:

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$$A_{h\Theta} = A_h \left(\mathrm{Cos}\Theta + \frac{\mathrm{Sin}^2\Theta}{2P_2}\right)$$

where

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$P_2 = \frac{S_r}{A}$
S_r = Radius from slew axis to outer edge of segment
A = Depth of advance = A_h Cosec α
α = Angle of frontal batter
Θ = Angle slewed from the direction of advance

The slew angle in the falling cut can be considerably more than that of the bench cut because $A_{h\Theta}$ does not become negligible until Θ exceeds 85°. Theoretical calculations will show that the slewing speed is to be increased in the same relation as before and that the output is identical by either of the two cuts. However, this conclusion can be erroneous if the material is not homogeneous in all the planes, which is usually the case with soils.

BWE OPERATIONS IN USA

Reference has already been made to the introduction of wheel excavators in Illinois strip mines during the 1940's. There are only a very few wheels in operation in the United States. Except for one operation each in the states of Washington and Wyoming, there are six wheel excavators in the Illinois coal fields. In American strip mining practice, bucket wheels have been used to remove loose top soil and soft beds, whereas the harder beds and the coal itself are handled by large stick shovels or draglines. In order to fully utilize the capacity of the two units in a tandem operation, the output capacity and dimensions of the wheel and shovels must be matched.

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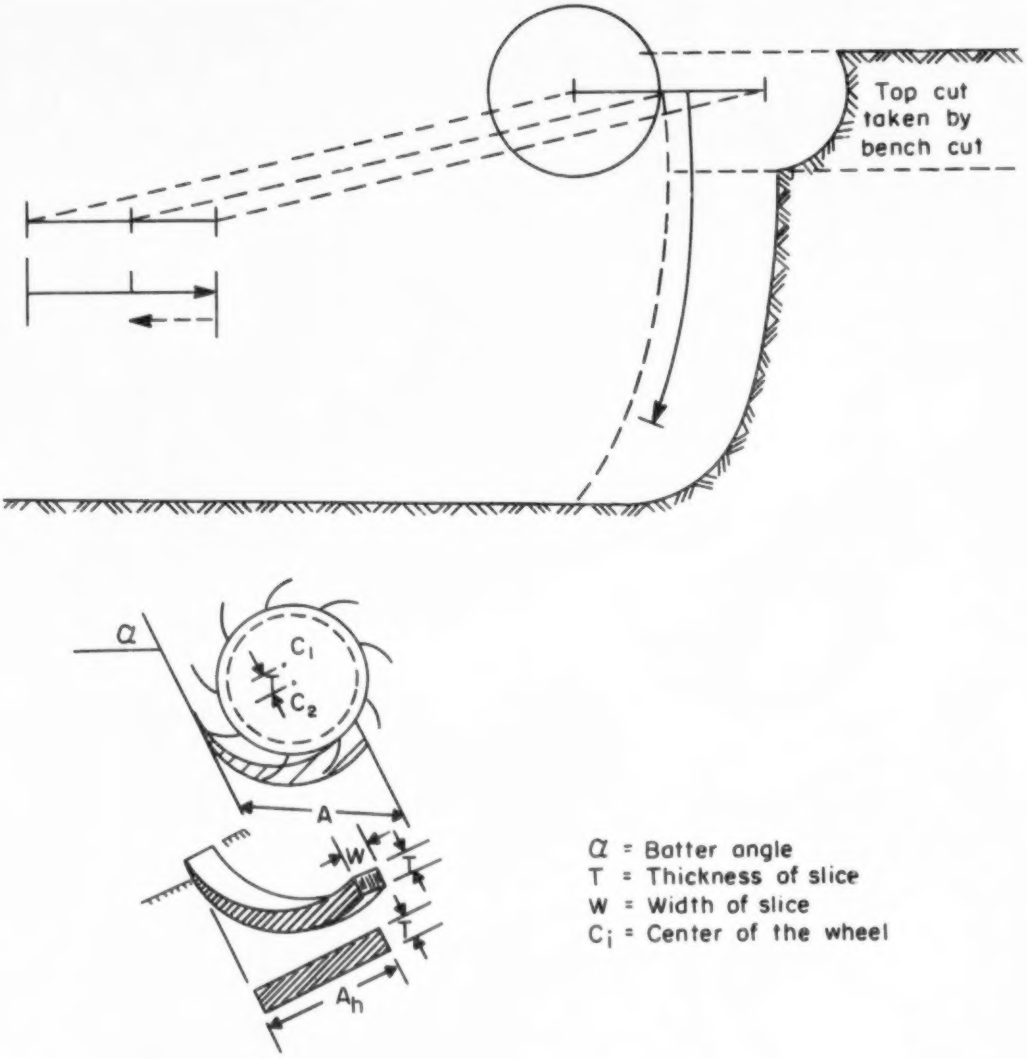
Top cut taken by bench cut
α
C₁
C₂
A
W
T
T
A_h

α = Batter angle
T = Thickness of slice
W = Width of slice
C_i = Center of the wheel

Figure 101. Fall cut block digging operation with BWE.

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The Kolbe wheel excavator can operate only by the frontal block digging method on benches of limited width. Overburden is stripped with the excavator operating in a long trench about 30 to 32 meters wide at the bottom and transferring it directly to the spoil heap. With burdens of 30 meters, the width of the cut is only about 15 meters (Figure 31). Since there will be occasions when the wheel and the tandem shovel have to cross each other, the pit must be designed wide enough to allow this.

BWE SELECTION PARAMETERS

The required data for input are divided into four categories:

1. Mining Dimensions: These include (a) the length of the mining property; (b) the number of benches; and (c) the elevations of the benches.
2. Soil Characteristics: The data required here are (a) the various soil types; (b) their specific cutting resistances; (c) the operating speed of the wheel in the soil types; (d) the bucket filling capacity; and (e) the swell factor.

In soils with higher specific cutting resistances, the wheel is speeded up and the successive slices that are taken are thinner. This seriously affects bucket filling capacities and in some cases less than 25 percent of the theoretical output is mined. Swell factors for sand and similar loose, broken material is not very significant, whereas a compacted material may increase in volume by 30 percent to 40 percent on fragmentation.

3. Wheel Specifications: The data input here include (a) the diameter of the wheel; (b) the number of buckets in the wheel; (c) the length of the slew axis; (d) the overall weight of the machine, and (e) the width of the machine.
4. Miscellaneous: Realistic efficiency figures are necessary for the mechanical and electrical components of the BWE to arrive at proper production figures.

Much research and trial-and-error experience have gone into the most recent designs of the digging wheel, its bucket and other appurtenances. The result is a device that will satisfactorily handle the variety of materials encountered, especially sticky clays, soils containing some floating boulders, and shale of varying hardness and composition. BWEs are machines designed for continuous digging and discharge. The continuous steady action requires less specific power and greatly reduces load fluctuations, characteristic of intermittent methods like that of shovels and draglines.

The range of application of BWE is not limited to mining alone. The classic method of stacking out and reclaiming bulk materials by conventional machines has been replaced today by far more economical use of conveyors, stackers, and bucket wheels.

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A BWE is an asset to land reclamation. Spoil piles are much more regular in size and contour than those produced by shovels or draglines. The hard rock is spoiled at the bottom of the mined area by the shovel or the dragline, and the original loose top soil is returned to relatively the same position, making replanting easier. In many cases, the crop and grazing value of the reclaimed spoil areas is greater than prior to excavation.

DRILLING AND BLASTING

Since this subject has been covered in detail in an earlier section, here the important considerations will be briefly outlined. Selection of drilling equipment depends on the types of materials and their strengths. Thus, penetration resistance to drilling should be studied before selection of a drill and a method of drilling. Soils and other loose materials do not require fragmentation, but are drilled in the process of drilling underlying consolidated materials, when these loose materials overlie the rocks. Although there are several methods of drilling, only mechanical methods which utilize percussion drills or roller bit rotaries, or a hybrid of both, are considered.

Selecting a particular drill involves the evaluation of energy and power consumed, bit wear and replacement cost, drilling rate, and cost of drilling.

This evaluation depends on several factors, namely: 1) the design and operating parameters comprised of drill, rod, and bit which should be selected to match the rock type; 2) lubrication and cleaning variables, i.e., circulation fluids usually either water, air, or mud; 3) the hole geometry, i.e., size and depth, which are exogenously determined; 4) environmental and geologic factors such as the rock properties, structural and petrological properties of the rock, as well as the overburden stress, all of which determine the drillability of the overburden; and 5) factors external to drilling such as cost of labor, job site, scale of operation, type and availability of power to the site, climatic conditions, and management and supervisory efficiency.

Selection of Drilling Equipment

Even after good mathematical analysis, the pit engineer has to use judgement to arrive at the selection of drilling equipment. The following procedure would aid in decision making:

1. Determination and specification of job conditions, i.e., labor, weather, and site location.
2. Determination or goal setting for the unit operations, e.g., excavation, haulage, crushing capacity, production required, pit geometry, fragmentation, and throw required.
3. Determination of number of drill holes, size, design, and area of influence expected.

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4. Determination of rock strength and drillability and selection of feasible drill methods.
5. Specification of operating criteria for each system to be considered, i.e., rock, bit, and circulation fluid.
6. Estimation of the performance characteristics including costs for depreciation, maintenance, power, and fluids.
7. Selection of the drilling system and equipment which best meet the above requirements as well as has the lowest overall cost (if possible). Table 46 gives information for selection of drill type based on rock strength and type.

Number of Drills

In surface coal mining, drilling and blasting is done for removal of the overburden. Amount of drilling and blasting depends on the types of strata present in the area. Hard rocks require more drilling and blasting, and often the soil is not fragmented. The number of drills needed for drilling can be calculated as follows:

SYE = Square yards of coal to be exposed
COB = Cubic yards of overburden removed
DS = Spacing of the drill holes, ft
DB = Burden of the drill holes, ft
DP = Drilling pattern $(\text{ft})^2$
LDPM = Length of drilling required per month (ft/month)
NH = Number of drill holes
DR = Drilling rate (ft/hour)
HPS = Hours scheduled per shift
SPD = Shifts per day
DPW = Drilling days per week
SPM = Weeks worked per month
SMH = Schedule monthly hours
DPM = Number of drills required
DP = DS x DB

$$\text{NH} = \frac{(\text{SYE})(9)}{\text{DP}}$$

$$\text{LDPM} = \left(\frac{\text{COB}}{\text{DP}}\right)(27)$$

$$\text{SMH} = \text{HPS} \times \text{SPD} \times \text{DPW} \times \text{WPM}$$

$$\text{DPM} = \frac{\text{LDPM}}{(\text{SMH})(\text{DR})}$$

EXPLOSIVES REQUIRED PER MONTH

The explosive commonly used in stripping is ANFO. Ammonium nitrate is not readily explosive until the addition of the fuel oil, which acts

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TABLE 46. APPLICATION OF DRILLING AND PENETRATION METHODS TO TYPES OF ROCK (PFLEIDER, 1968)

	ROCK TYPE			
RATING → CATEGORY → EXAMPLES →	1 Soft (shale, weath. limestone)	2 Medium Hard (limestone, weath. sandst.)	3 Hard (granite, chert)	4 Very Hard (quartzite, taconite)
DRILLING METHOD ↓				
rotary, drag	x	x		
rotary, roller	x	x	x	x
percussion		x	x	x
jet piercing			x	x

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like a catalyst to enhance its explosive strength. It is much safer, less expensive, easier to prepare and more effective for rock blasting than most comparable explosives. ANFO is available in bulk premixed, on-spot mix and bagged forms.

Fundamental to effective blasting is the use of correct amount of explosive for the volume of overburden to be fragmented. This relationship is usually expressed as the "powder factor," which is defined as the ratio of cubic yards of overburden fragmented per pound of explosives. It varies from 2.5 to 9.0 (from hard rocks to soft soils).

To determine the amount of explosive needed per foot of hole, the area of influence of a drill hole (DP) is divided by 27, and then divided by the powder factor "PK".

$$\left(\frac{DP}{27}\right) \cdot \left(\frac{1}{PK}\right)$$

Therefore, knowing the number of holes drilled per month, the drilling pattern and the overburden depth, the amount of explosive required per month (AANFO) can be calculated:

DC = Length of explosive charge in hole in feet
AANFO = Amount of explosive required per month

$$AANFO = \left(\frac{DP}{27}\right) \cdot \left(\frac{1}{PK}\right) \cdot (NH) \cdot (DC) \text{ lb}$$

Alternatively, if the entire overburden is to be blasted, then

$$AANFO = \frac{COB}{PK}$$

where

COB = Cubic yards of overburden removed

But it is common knowledge in stripping, that not all the overburden requires fragmentation before removal. This is particularly true for topsoil and soft soil. Therefore, actual consumption of explosive must be calculated on the basis of experience with the overburden, i.e., for individual application, the fraction of overburden that should be fragmented must be known.

COAL LOADING

Although there are several methods of coal loading, the method considered in this report is a shovel-truck combination.

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Here, shovels are selected to match the advance of the stripping equipment and the capacity of the trucks. Even when matching is achieved, the general trend is to select a loader with a slightly higher loading ability than calculations show for normal operations. The calculations for selection of a loader are as follows:

CTL = Loader cycle time in seconds
BL = Bucket size of loading shovel in cu yd
TPB = Tonnage per bucket
F = Factor for converting 1 cu yd of loose coal to tons
SMH = Scheduled monthly hours
SMH = HPS x SPD x DPW x WPM
NPH = Number of cycles

$$NPH = SMH \times \frac{3600}{CTL}$$

$$TPB = \left(\frac{P}{1 - A}\right) \div NPH$$

$$BL = \frac{TPB}{F}$$

A loader with this capacity could easily handle the monthly production.

Truck Fleet Selection

The truck capacity and loader capacity should be matched so that the truck waiting time on loader and loader waiting time for trucks are minimized. The calculations for matching the loader with the truck and for selecting the required number of trucks are as follows:

HD = Hauling distance in ft
FT = Fixed time in min
DT = Delay time in min
MT = Maneuver time in min
TLK = Truck loading time in min
CTK = Truck cycle time in min
TKC = Truck capacity in tons
TPB = Tonnage per bucket
NT = Number of trucks required

$$TLK = \left[\frac{TKC}{TPB}\right] \left[\frac{CTL}{60}\right]$$

$$FT = DT + MT + TLK$$

$$CTK = Ft + \left[\frac{HD}{(S1)(88)} + \frac{HD}{(S2)(88)}\right]$$

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$$NT = \frac{CTK}{TLK} + 1$$

The travel time and return time of the truck used above should be calculated using average velocities for loads (S1) and empties (S2). Figure 102 will aid in the sizing of trucks with regard to shovel bucket capacity.

SECONDARY EQUIPMENT SELECTION

The secondary or support equipment in surface mining ranges from highly sophisticated construction equipment to small farm machines for reclamation and revegetation purposes. In this section, the support equipment considered are bulldozers, scrapers, and front-end loaders. These machines are used for initial land clearing, mine road construction, rehandling of spoil, loading, and haulage and reclamation jobs. They are virtually indispensible in tight spots where larger machines cannot easily maneuver.

As with shovels and draglines, the secondary equipment are associated with fixed and variable components in their cycle times. For example, Atkinson (1971) gives the following values for the fixed and variable components of the cycle time for front-end loaders (Table 47 and Figure 102). According to Caterpillar Tractor Company (1975), a fixed time of 0.40 minutes (24 seconds) for front-end loaders working in loose granular material on a hard-smooth operating surface is recommended. This fixed time includes load, dump, four reversals of direction, full cycle of hydraulics and minimum travel.

Since material type affects the fixed time of the loader, Caterpillar Tractor Company (1975) recommends the corrections given in Table 48 for other material characteristics and operating conditions.

For scrapers, Caterpillar (1975) recommends the use of 0.7 minutes (42 seconds) each for loading, maneuvering, and spreading.

There are several methods by which the secondary equipment can be selected. They should match the performance of the primary stripping equipment. Two alternative selection methods are outlined here:

A. Procedure I. Briefly, this procedure consists of the following steps:

1. Select a suitable bucket (or blade) size. The capacity is usually expressed in cubic yards.
2. Calculate the variable part of the cycle time.
3. Determine the fixed times from time study. Sometimes, this can be obtained from manufacturers' handbooks.
4. Calculate the total cycle time by adding the variable cycle time to the fixed cycle time.
5. Calculate the number of passes per hour.
6. Determine the hourly production by multiplying the number of passes (cycles per hour) by load per cycle (capacity).

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TABLE 47. FIXED TIMES FOR FRONT-END LOADERS (ATKINSON, 1971)

BUCKET CAPACITY	FIXED TIME		SEC
yd^3	Easy Digging	Medium Digging	Medium-Hard Digging
5	32	33	41
6	33	34	42
7	33	35	44
10	37	39	51
12	39	42	56
15	41	44	60

Note: The application of front-end loaders in hard digging conditions is marginal and requires comprehensive investigation.

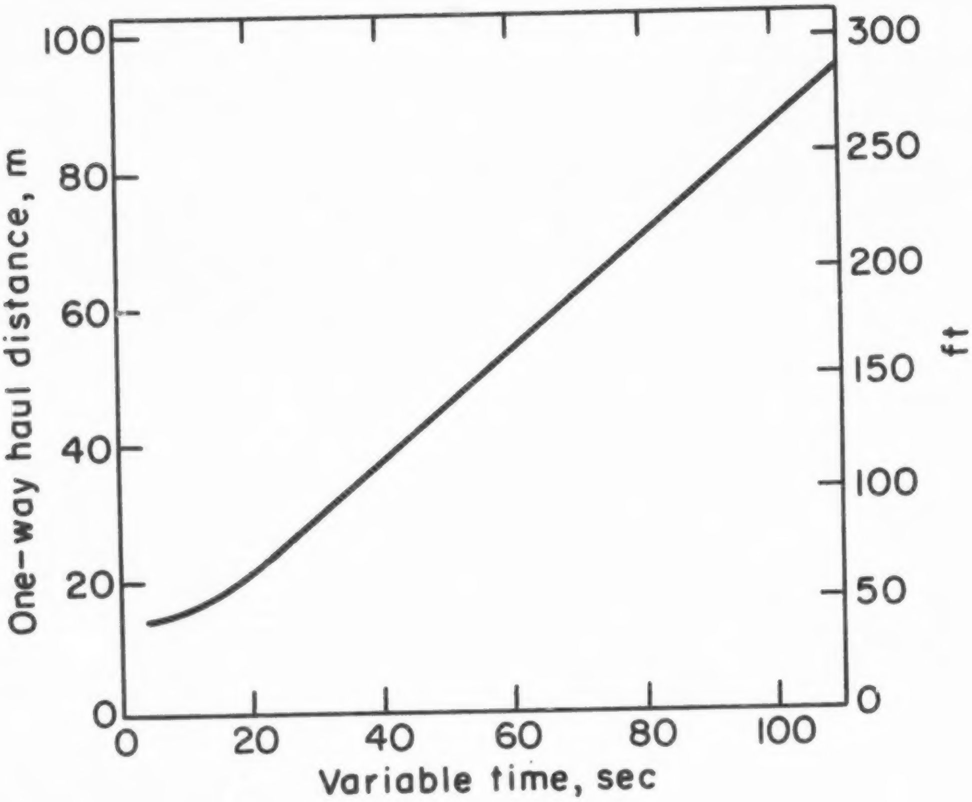

Figure 102. Front-end loaders variable time (Atkinson, 1971).

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TABLE 48. MISCELLANEOUS CORRECTION FACTORS (CATERPILLAR, 1976)

MATERIALS	MINUTES TO BE ADDED TO 0.4 MINUTES
Mixed	+ 0.02
Up to 1/8 inch	+ 0.02
1/8 inch to 3/4 inch	- 0.02
6 inches and over	+ 0.03 and up
Bank or Broken	+ 0.04 and up
Conveyor or Dozer piled 10 feet and up	0.00
Conveyor or Dozer piled 10 feet or less	+ 0.01
Dumped by truck	+ 0.02
Management Conditions	
Common ownership	up to - 0.04
Independently owned trucks	up to + 0.04
Constant operation	up to - 0.04
Inconsistent operation	up to + 0.04
Small target	up to + 0.04
Fragile target	up to + 0.05

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To arrive at the actual production, correction factors from the manufacturer's handbook or from experience are applied.

B. Procedure II. Here, one can use the maximum production per hour given by the manufacturer for some standard condition. To determine the actual production, job condition correction factors must be applied to the maximum production.

Correction Factors. Most of the correction factors discussed earlier for draglines and shovels equally apply to the secondary equipment. Additional factors that apply to secondary equipment are discussed here. Some of these factors are presented in Tables 49 and 50.

Material Factor. The performance of bulldozers, scrapers and front-end loaders are affected by material constituents and characteristics. Since these equipment have varied applications, from loading and grading in loose material to excavating hard frozen grounds, their capacity for production as given by manufacturers must be corrected by a factor known as material factor. Table 50 gives such factors for bulldozers.

Grade Factor. Since secondary equipment are mobile, their production capability is affected by grade of the roadway and the work area conditions. Factors are applied to production capabilities as given by manufacturers to correct for the grade effect.

Visibility Factor. Impairment of operator's visibility due to atmospheric conditions such as snow, fog, rain, etc., affect the performance of the secondary equipment. Visibility factors are applied particularly to bulldozers and scrapers.

Transmission Factors. This factor accounts for the transmission mode of the equipment; direct or automatic drive. The effect of the transmission train mode on the equipment's productivity can be corrected by applying transmission factors to the production capability given by manufacturers.

Dozing Factor. This factor accounts for the mode of operation of a bulldozer, whether slot dozing or side-by-side dozing. It applies to bulldozers only and relevant values are given in Table 49. Factors for operating efficiency and job efficiency, as applied to primary equipment, are also applied to secondary equipment. Values for these factors are given in Table 49.

1. Bulldozers. Bulldozers are used extensively in surface mining. They are extremely useful for dozing material through short distances under severe conditions. Selection of a suitable bulldozer can be accomplished by calculating,

Bb = Bucket or blade size (machine capacity) in cu yd

The choice of blade type depends on the kind of job application and soil conditions. Table 51 shows bulldozer blade selection chart according to job application.

TABLE 49. BULLDOZER CORRECTION FACTORS (CATERPILLAR, 1976)

Job Condition Corrections:	Track-Type Tractor	Wheel-Type Tractor
OPERATOR - Excellent	1.00	1.00
Average	0.75	0.60
Poor	0.0.60	0.0.50
MATERIAL -		
Type-		
Loose stockpile......................	1.20	1.20
Hard to cut; frozen-		
with tilt cylinder................	0.80	0.75
without tilt cylinder.............	0.70	----
cable controlled blade............	0.60	----
Hard to drift; "dead" (dry, non-cohesive material or very sticky material)............	0.80	0.80
Rock, ripped or blasted.............	0.60-0.80	----
SLOT DOZING..........................	1.20	1.20
SIDE BY SIDE DOZONG..................	1.15-1.25	1.15-1.25
VISIBILITY - Dust, rain, snow fog or darkness.....................	0.80	0.70
JOB EFFICIENCY - 50 min/hr.	0.84	0.84
40 min/hr.	0.67	0.67
DIRECT DRIVE TRANSMISSION.............		
(0.1 min. fixed time)................	0.80	----
*BULLDOZER - Angling (A) blade.........	0.50-0.75	----
Cushioned (C) blade..................	0.50-0.75	0.50-0.75
D5 narrow gauge...........	0.90	----
Light material U-blade (coal)..........	1.20	1.20
Blade bowl (stockpiles)............	1.30	1.30

*Note: Angling blades and cushion blades are not considered production dozing tools. Depending on job conditions, the A-blade and C-blade will average 50-75% of straight blade production.

TABLE 50. SWELL FACTOR - BULLDOZED MATERIAL (ATKINSON, 1971)

MATERIAL	SWELL FACTOR
Broken rock	1.65
Heavy clay (wet)	1.45
Earth with boulders	1.35
Earth	1.25
Sand and small gravel	1.1

TABLE 51. BULLDOZER SELECTION CHART FOR RIPPING AND DOZING APPLICATION (SKELLY AND LOY, 1975)

Job	Ground Condition	Rubber Tired Dozers			Track Dozers			Remarks
		Small	Medium	Large	Small	Medium	Large	
Clean up Around Small Shovel	Friable	x			x			No Ripping
			x					Medium Ripping
						x		Hard Ripping
	Hard Friable	x			x			No Ripping
			x			x		Medium Ripping
				x		x		Hard Ripping
	Hard Jagged				x			No Ripping
						x		Medium Ripping
							x	Hard Ripping
Clean up Around Large Shovel	Friable		x					No Ripping
			x					Minimum Ripping
			x			x		Medium Ripping
	Hard Friable		x					No Ripping
			x			x		Medium Ripping
				x		x		Hard Ripping
	Hard Jagged					x		No Ripping
						x		Medium Ripping
							x	Hard Ripping
Waste Dump Dozing Capabilities	Friable and Hard Friable	x						9 (26 Ton Trucks)
			x					32 (59 Ton Trucks)
				x				>32 (50 Ton Trucks)
	Hard Jagged				x			9 (26 Ton Trucks)
						x		32 (59 Ton Trucks)
	Wet Muddy				x			90 to 180 Tons/Hour
						x		270 to 720 Tons/Hour
							x	>720 Tons/Hour

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FT = Fixed time in min
TF = Travel loaded time in min
TE = Travel empty time in min
VT = Variable time in min
MT = Maneuver time in min
CT = Cycle time in min
LT = Loading time in min
NPH = Number of passes per hour
PPH = Production per hour (theoretical) in cu yd
MAXPH = Actual Production per hour in cu yd
HD = Haul distance in ft
S1 = Loaded speed in mph
S2 = Empty speed in mph
Bb = Bucket size of bulldozer in cu yd
LF = Load factor
BF = Bucket fill factor
AO = Operating factor

The selection procedures are outlined here.

6-1/2"

$$FT = MT + LT$$

$$TF = \frac{HD}{(S1)(88)}$$

9-1/8"

$$TE = \frac{HD}{(S2)(88)}$$

$$VT = TF + TE$$
$$CT = TF + FT \text{ (minutes)}$$

$$NPH = \frac{60}{CT} \text{ (cycles per hour)}$$

$$PPH = (NPH)(Bb)$$

$$MAXPH = (PPH)(LF)(BF)(AO)\left(\prod_{i=1}^{k} CF_i\right)$$

where k is the number of other applicable correction factors from the manufacturer's handbook, and CF_i is the value for the i^{th} correction factor. Alternatively, the actual production can be calculated as,

$$MAXPH = (MAXPHG)(LF)(BF)(AO)\left(\prod_{i=1}^{k} CF_i\right)$$

where MAXPHG is the maximum production obtained from catalog or handbook under specified conditions.

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2. Scrapers. There are several types of scrapers, namely, elevating and conventional scrapers, and single- and double-engine powered scrapers to mention a few. The major distinction is between wheel-tractor scrapers and crawler-tractor scrapers. Wheel scrapers, because of their mobility, are used in surface coal mining for topsoil removal, road building, etc. Shown in Figure 103 are the scraper cycle times as functions of distance and grade.

Using the same parameters as before and denoting the bucket capacity by "BS", the following relationship can be established. The calculations for the cycle time and the number of passes of cycles per hour are the same as for the dozers,

BS = Bucket size of scraper in cu yd
PPH = (NPH)(BS)

$$MAXPH = (PPH)(LF)(BF)(AO)(\prod_{i=1}^{k} CF_i)$$

Alternatively,

$$MAXPH = (MAXPHG)(LF)(BF)(AO)(\prod_{i=1}^{k} CF_i)$$

where k is the number of other applicable correction factors from the manufacturer's handbook and CF_i is the value if the i^{th} correction factor. For calculating the number of pusher tractors, the pusher cycle time must be known. The maximum number of scrapers that can be served by one pusher tractor is given by rounding the following ratio to the next lower integer,

$$\text{Number of scrapers per pusher tractor} = \frac{\text{Scraper cycle time}}{\text{Pusher cycle time}}$$

$$\text{Number of pusher tractors} = \frac{\text{Total number of scrapers}}{\text{Number of scrapers per pusher}}$$

3. Front-end Loaders. In selecting front-end loaders, one must avoid overestimating the capacity of the bucket. Improper bucket-sizing may lead to instability of the equipment. There are three Society of Automotive Engineers (SAE) ratings for front-end loader bucket capacities: a) struck capacity; b) heaped or rated capacity; and c) static tipping load.

BL = Bucket capacity in cu yd
DP = Dumping time in min
MS = Maneuvering time in min
Ft = Total fixed time in min
LT = Loading time in min
CT = Cycle time in min
MAXPH = Actual production per hour in cu yd
LF = Load factor
BF = Bucket fill factor
AO = Operating factor
NPH = 60/Ct
PPH = (BL)(NPH)

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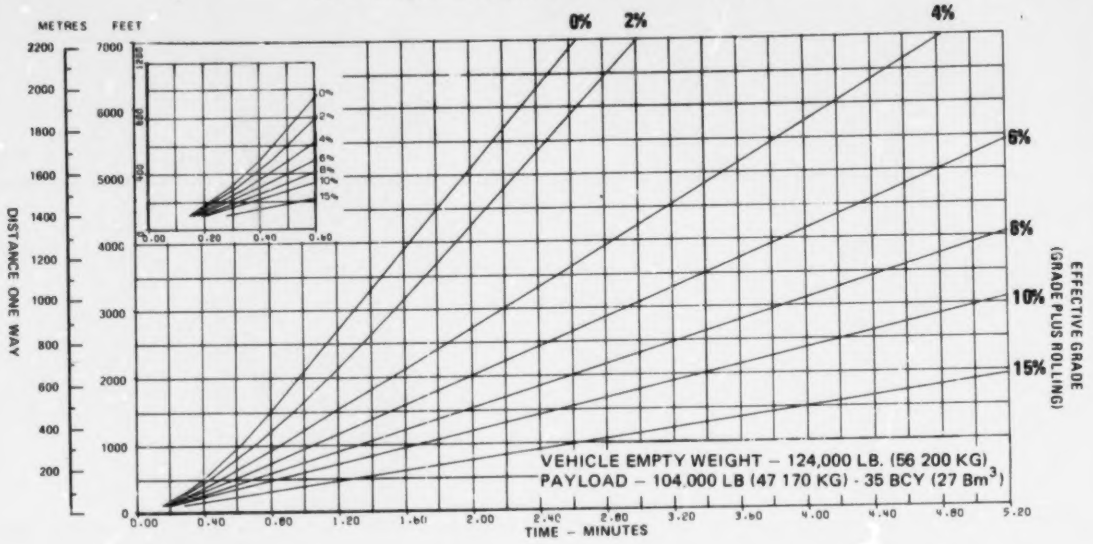
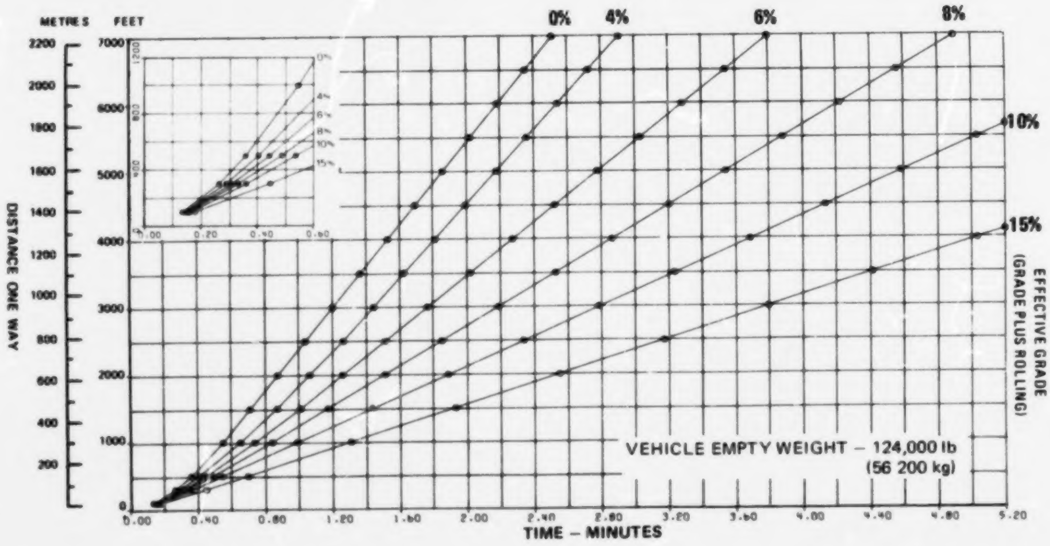


651B (37.50 X 39) DISTANCE VS TIME – LOADED



651B (37.50 X 39) DISTANCE VS TIME – EMPTY

Figure 103. Scraper distance versus travel times (Caterpillar, 1976).

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$$\text{MAXPH} = (\text{PPH})(\text{LF})(\text{BF})(\text{AO})\left(\prod_{i=1}^{k} \text{CF}_i\right)$$

where k is the number of other applicable correction factors from the manufacturer's handbook and CF_i is the i^{th} correction factor.

Alternatively, Table 52 provides manufacturer production capacities for front-end loaders, from which actual production can be calculated by application of the correction factors,

$$\text{MAXPH} = (\text{MAXPHG})(\text{LF})(\text{BF})(\text{AO})\left(\prod_{i=1}^{k} \text{CF}_i\right)$$

In using the correction factors average conditions should be assumed where performance data is not available. Some correction factors are a combination of two or more other factors. For example, "AO," the operating factor, is a product of operator efficiency factor and job management factor. Where accurate data are available, results would reflect the actual performance.

6-1/2"

SUMMARY

This chapter has briefly reviewed the engineering considerations in equipment selection and pit design. Equipment selection formulas have been provided to select the bucket size, reach of machines, width of pit, number of drills, loading shovels, trucks, and bulldozers to achieve a production target. Formulas have also been provided to select bucket-wheel excavators, scrapers, and front-end loaders. Important job correction factors were discussed. In almost all cases, the selection of other equipment is dictated by major stripping equipment; because of this, great care must be exercised in their selection.

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TABLE 52. WHEEL-LOADER PRODUCTION ESTIMATING TABLE (CATERPILLAR, 1976)

CUBIC YARDS/60 MIN. HR.
ESTIMATED BUCKET PAYLOAD - BANK CUBIC YARDS

Estimated Cycle Times		Bucket Sizes										
Hundredths	Cycles Per Hr.	1.0	1.5	2.0	2.5	3.0	3.5	4.0	4.5	5.0	5.5	6.0
.35	171											
.40	150	150	225	300	375	450	525					
.45	133	133	200	268	332	400	466	530	600	665	730	800
.50	120	120	180	240	300	360	420	480	540	600	660	720
.55	109	109	164	218	272	328	382	436	490	545	600	655
.60	100	100	150	200	250	300	350	400	450	500	550	600
.65	92	92	138	184	230	276	322	368	416	460	505	555
.70	86							342	386	430	474	515
.75	80											

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TABLE 52. (continued)

CUBIC YARDS/60 MIN. HR.
ESTIMATED BUCKET PAYLOAD - BANK CUBIC YARDS

Estimated Cycle Times		Bucket Sizes							
Hundredths	Cycles Per Hr.	6.5	7.0	7.5	8.0	8.5	9.0	9.5	100
.35	171								
.40	150								
.45	133	865							
.50	120	780	840	900	960	1003	1080	1140	1200
.55	109	705	765	820	870	925	980	1008	1090
.60	100	650	700	750	800	850	900	950	1000
.65	92	600	645	690	735	780	830	875	920
.70	86	560	600	645	690	730	775	815	860
.75	80		560	600	640	680	720	760	800

JOB EFFICIENCY	EFFICIENCY FACTOR	BUCKET LOAD FACTOR
Worktime/Hr.		
60 Min. Hr.	100%	Bucket Size X 1.00
55	91%	.95
50	83%	.90
45	75%	.85
40	69%	.80
--	---	.75

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SECTION 9

RECLAMATION

INTRODUCTION

Competition for land to be used for agriculture, living space, recreation, and industry is becoming more intense. Mineral exploration, development, extraction and processing all have different impacts and conflict with these essential uses. The very nature of the non-renewability of a mineral deposit necessarily makes mining a temporary activity, to be completed sooner or later. The importance of mining, particularly surface mining, however, can hardly be overemphasized as it is doubtful if there is any aspect of human life which does not require the products of mining. It is known that over 90% of the mined products are obtained through the surface method. In fact, more than 75% of the land affected by surface mining is the result of the extraction of coal, crushed stone, and sand and gravel. Therefore, it is necessary to be concerned with the concurrent and sequential land use for optimum land utilization. The allocation of land for agriculture, forest, wildlife, urban growth, recreation, transportation, and so forth, is compounded by political, economical, sociological, and ecological considerations. When some of these lands hold mineral values, the big question is: How can one mine these riches without impairing the utility value for satisfying land demands during and after mining? While this is too frequently interpreted as a problem, it should also be recognized as an opportunity to remold the surface for a desirable end-use.

LAND EFFECTS

Even though surface mining of coal constituted only 0.3% in 1914 and 7% in 1937 of the total U.S. coal production, the use of surface mined lands for other productive purposes was demonstrated and pursued by some coal companies voluntarily as early as 1920. In fact, prior to 1965, there were only seven states which had specific laws requiring land reclamation with West Virginia enacting the first legislation in 1939. The post-1965 period has been one of rapid growth in laws and regulations with regard to environmental control in general, and surface mining in particular, both at the federal and state levels. This discussion of land effects is provided only to serve as an overview of the genesis of the environmental problems associated with surface mining.

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Land disturbances from surface mining, in terms of area affected, have grown to enormous proportions in recent years, for the following technological reasons: as easily mined and richer deposits are fast being extracted, and in some cases, completely exhausted, attention has been directed towards mining larger tracts of lower-grade, higher cost deposits; in surface mining of coal, the ratio of overburden thickness to that of coal has been increasing; and current operations are involved with handling more material, and often from greater depths, than heretofore. The need to exploit vast reserves with thick cover and heavy relief has resulted in the development of larger and more powerful equipment which can displace and remove enormous amounts of materials at a much faster rate.

Land damages that are cited against surface mining are mainly due to the destruction of surface topographies and of soil conditions that existed before mining commenced. Often, the potential productivity of the soil for plant growth is greatly reduced after mining. Soils that are disrupted by these operations are often chemically active and toxic, thereby becoming a source for water pollution. Also, if the overburden is a massive rock formation, huge blocks of rock occur in the graded spoil which make it difficult for the smooth passage of farm machinery. Much larger areas are also affected by the unconsolidated spoil heaps and voids because these conditions affect drainage patterns. Here, the natural processes of erosion and sedimentation are accelerated, moving huge volumes of soil into receiving streams. When not properly graded and maintained against dumping waste and refuse, these sites become potential health hazards. Finally, the steep walls created as a result of thick burdens and heavy relief, coupled with the possibility of ground movement with lapse of time, present a safety hazard in inhabited neighborhoods and are aesthetically unattractive. Additional areas may also be affected by the associated haulroads and preparation and processing plants.

A summary of the major environmental effects due to the various surface mining unit operations is presented in Table 53. A detailed report on this subject has been prepared by Grim and Hill (1974). While this section is concerned with surface mining effects on land and land reclamation, the effects on water, air, wildlife, and other resources must also be considered. In fact, all these effects are not mutually exclusive. For example, air pollution may eventually lead to water pollution and both may degrade the land. Surface mining without control may result in serious blights and hazards to public health, contaminate air and water resources, adversely affect land values, create public nuisances, and generally interfere with community development. In fact, the controversy with regard to surface mining of coal is so intense that some have sought a complete ban on surface mining of coal and a greater emphasis on underground mining.

RECLAMATION PRACTICES

Rehabilitation of surface-mined lands is, however, a tangible program. In fact, strip-mined land, abandoned open pits, and spoil and waste piles are being shaped and graded, stabilized and seeded, and returned to productive use with proper care and management. Mined-area spoil banks that can

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TABLE 53. POTENTIAL MAJOR EFFECTS OF SURFACE MINING UNIT OPERATIONS ON LAND, WATER AND AIR RESOURCES

Surface Mining Unit Operations	Land and Soil									Water							Air					Wild Life								Other	
	Soil Erosion	Overburden Swelling	Toxic Strata	Soil Inversion	Soil Stability	Landslides	Spoil Piles	Oil Spills	Coal Spills	Aquifer Effects	Runoff Alteration	Sediments	Toxic Substances	Groundwater Contamination	Industrial Effluents	Sludge	Exhaust Emissions	Dust	Noise	Other (Welding, etc.)	Blasting Fumes	Habitat Altered	Species Diversity	Aquatic Life	Animal Essentials	Accident/Deaths	Soil Organisms	Vegetation Potential	Wildlife Disturbed	Aesthetic	Dangerous Material
1. Exploration	x									x							x	x	x										x		
2. Area Dewatering Diversion, Etc.	x										x	x	x	x								x	x	x	x		x		x		
3. Drilling										x	x						x	x	x										x		
4. Blasting		x	x			x				x	x							x	x		x	x				x			x	x	x
5. Stripping (Overburden Removal)			x	x	x		x	x									x	x	x	x		x					x			x	
6. Haulage	x							x	x			x					x	x	x			x				x	x		x		
7. Top Soil or Other Soil Storage	x		x				x					x		x				x				x					x	x	x	x	
8. Maintenance								x									x	x	x	x										x	x
9. Beneficiation													x	x x	x	x		x	x											x	

be revegetated present minor problems and have great potential for development. Various reclamation programs that are being actively pursued include returning the ground for agricultural and livestock farming, for reforestation, for recreation, and for housing and industrial sites (Figures 104, 105, 106, 107, 108, 109). The possibilities for development under these conditions are limited only by cost benefit and innovation. There are, however, marginal and problem spoils (acid, toxic, etc.) which require special attention and even better planning. Concern for good mining practices and effective land reclamation programs is witnessed by the adoption of new federal regulations and an increased research and development effort.

On August 3, 1977, the U.S. Congress enacted Public Law 95-87 titled The Surface Mining Control and Reclamation Act of 1977. It represents a significant effort on the part of the Federal Government to establish a nationwide program to protect society and the environment from the adverse effects of surface mining operations. Among the stated purposes of the 1977 Act, the following are particularly relevant to mine planning and operation:

1. To assure that surface coal mining operations are so conducted as to protect the environment.
2. To assure that adequate procedures are undertaken to reclaim surface areas as contemporaneously as possible with the surface coal mining operations.
3. To assure that the coal supply essential to the Nation's energy requirements and to its economic and social well-being is provided, and strike a balance between protection of the environment and agricultural productivity and the Nation's need for coal as an essential source of energy.

Even without the federal law, new or amended surface mine regulations were being enforced in 38 states. At the present time, several states are revising their statutes to be in accordance with the federal law.

Currently 99% of the nation's surface mined coal is being produced under land reclamation programs (Daley, 1974). In the history of the U.S., surface mining has disturbed 0.177 percent of the land and about one third of that has been judged as reclaimed by the U.S. Department of Interior.

In 1972, there were over two million acres of land requiring reclamation, and about two million not requiring reclamation for a total of 4,004,900 acres of disturbed land.

In the seven-year period between 1965 and 1971 private landowners reclaimed over a third of a million acres. More than 90 percent of the four million acres disturbed up to 1972 is privately owned. The USDA Soil Conservation Service (SCS) began participating in land restoration in the 1930's. Soil Conservation Service is active in developing new plants that can survive under the difficult slopes and acid conditions found on

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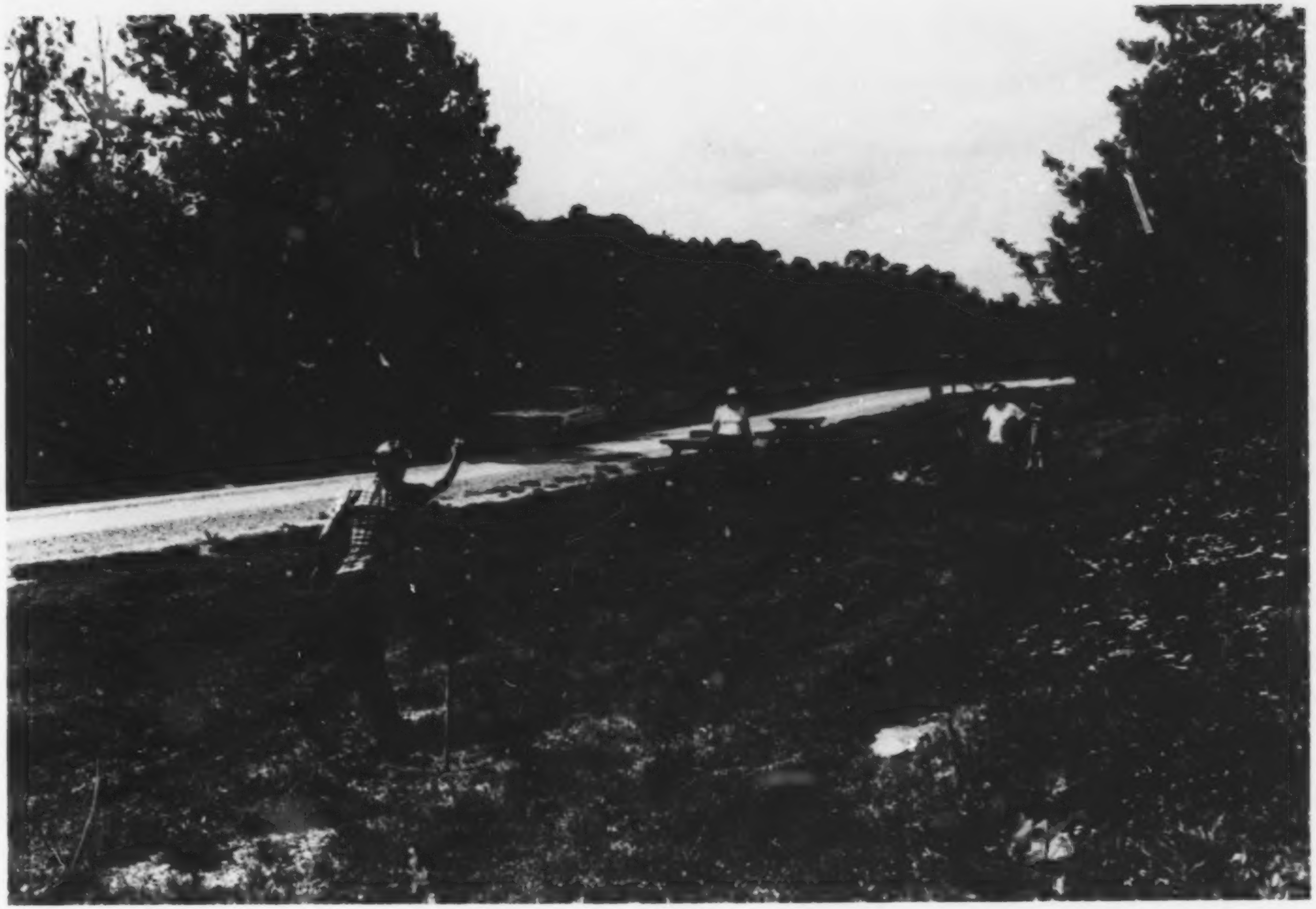

Figure 104. On land reclaimed with grass and trees after strip mining, responsible coal companies have created public recreation areas like this roadside park and campground in southeastern Ohio.

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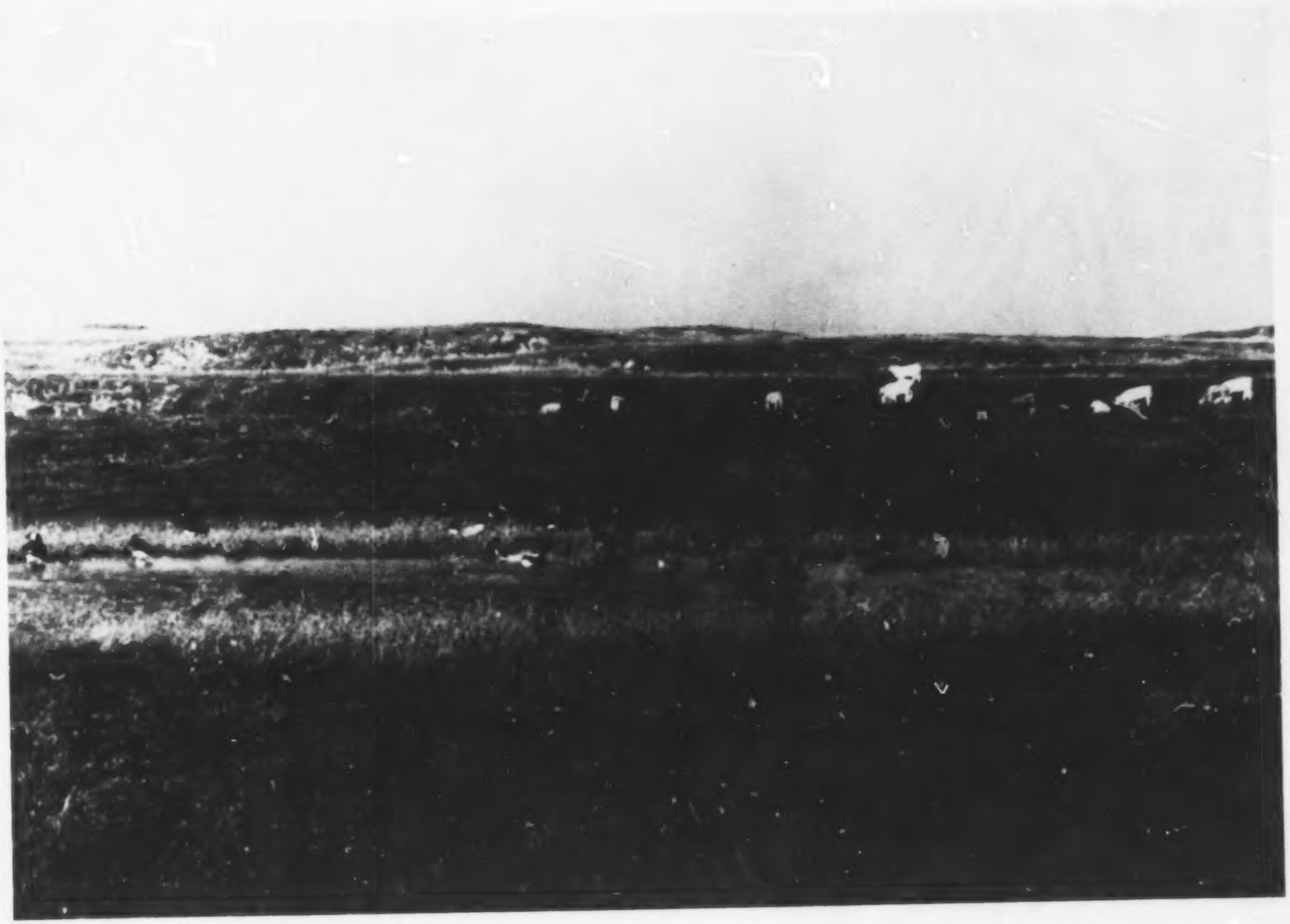

Figure 105. Charolais cattle and Giant Canadian Geese on mined lands.

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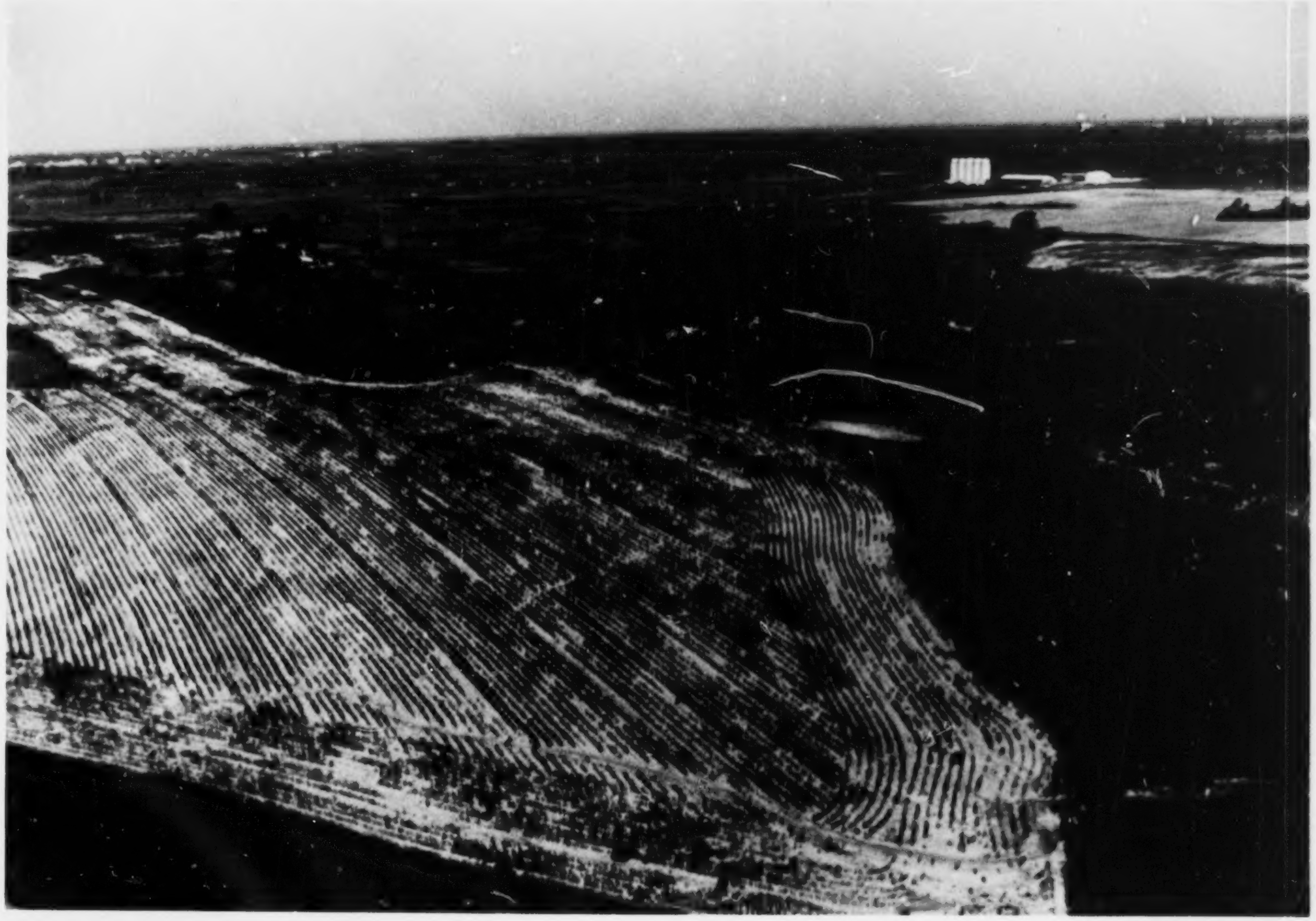

Figure 106. Corn, as well as forage, can grow on former coal land. The ponds in the background (former pits) provide watering places for cattle.

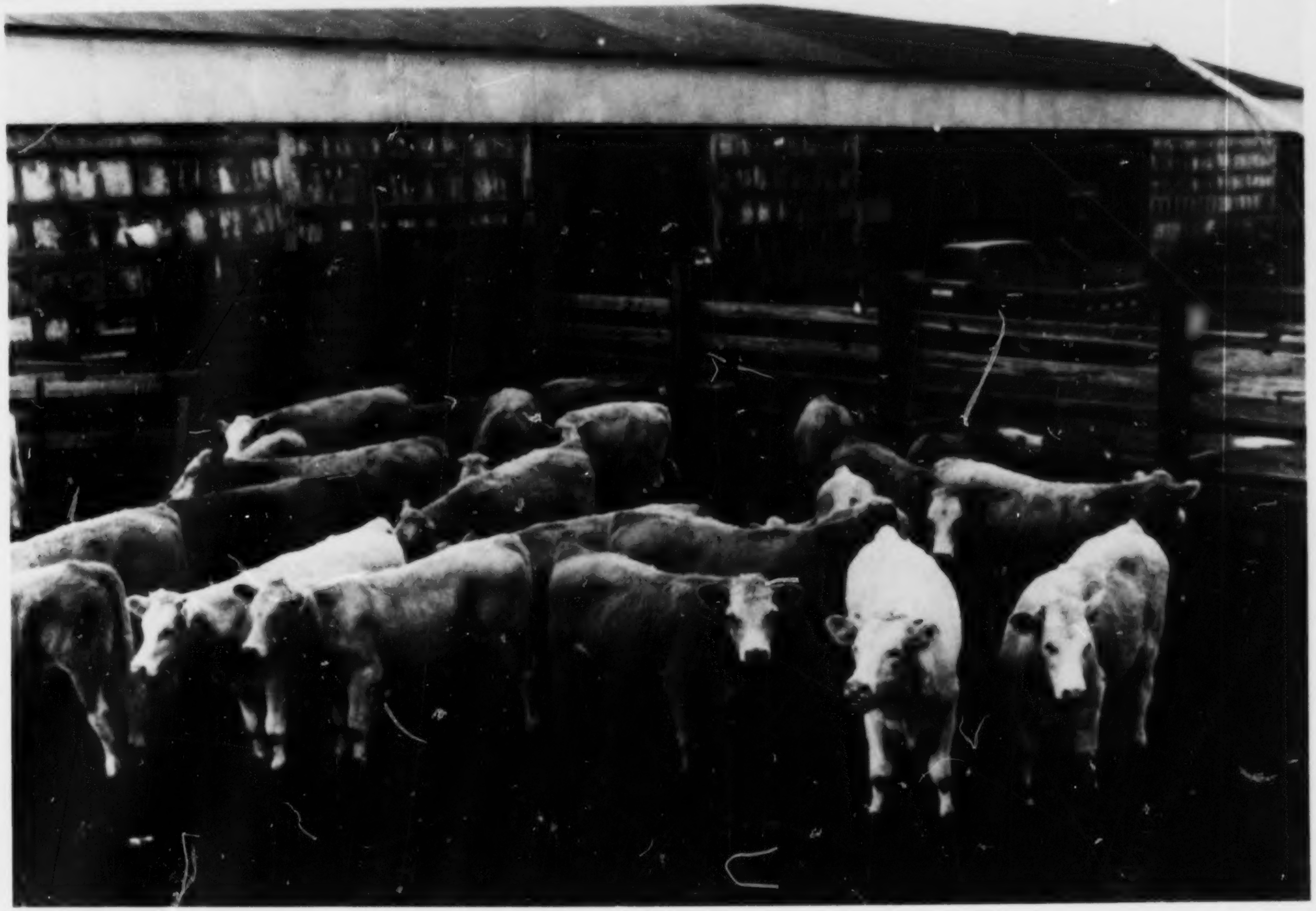

Figure 107. Charolais calves fatten in a feedlot on strip-mined land in Indiana. The farm machinery shed in the background was a repair shop for mining equipment in the days when the area was strip mined.

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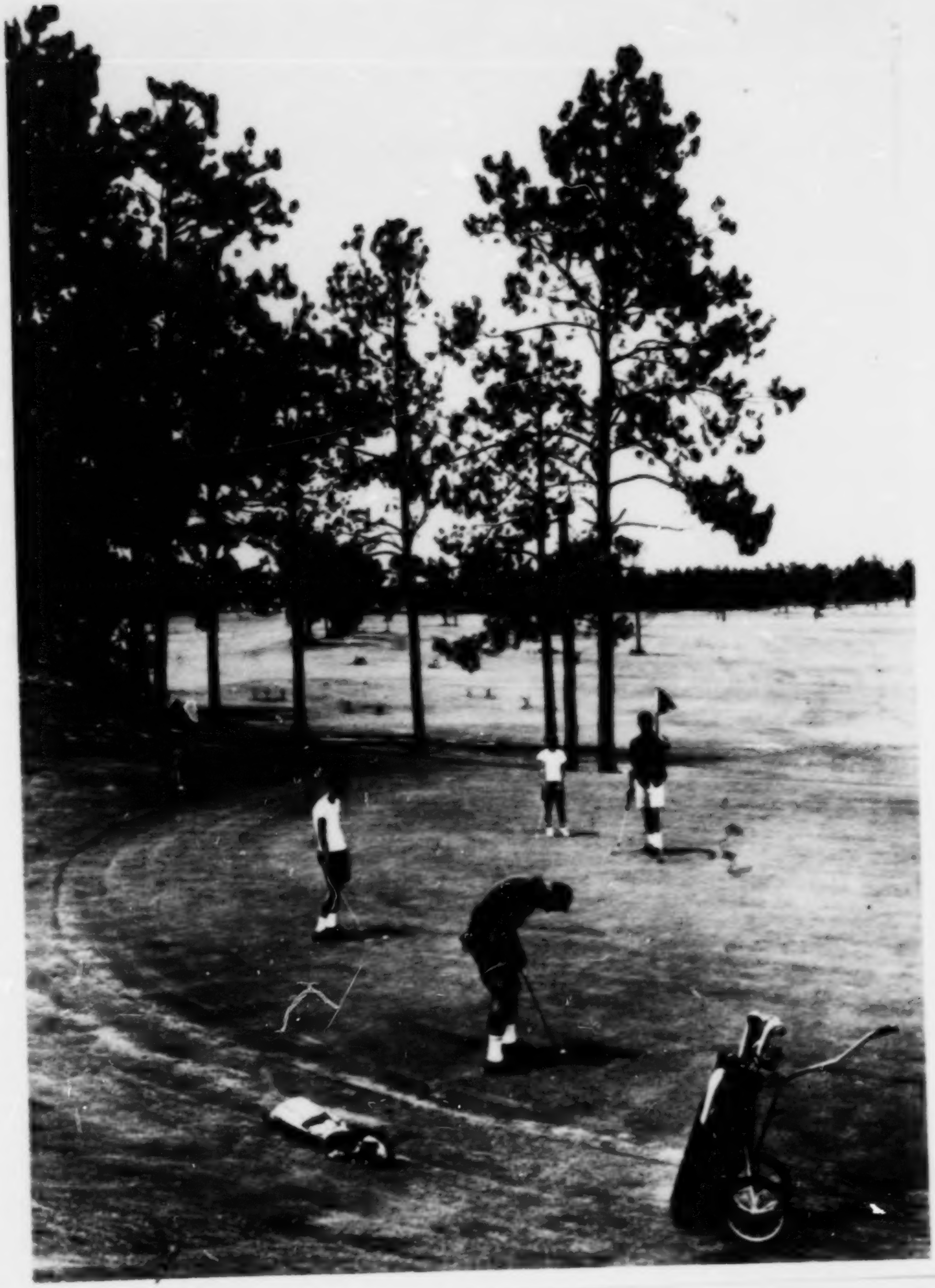

Figure 108. A golf course on strip mined land.

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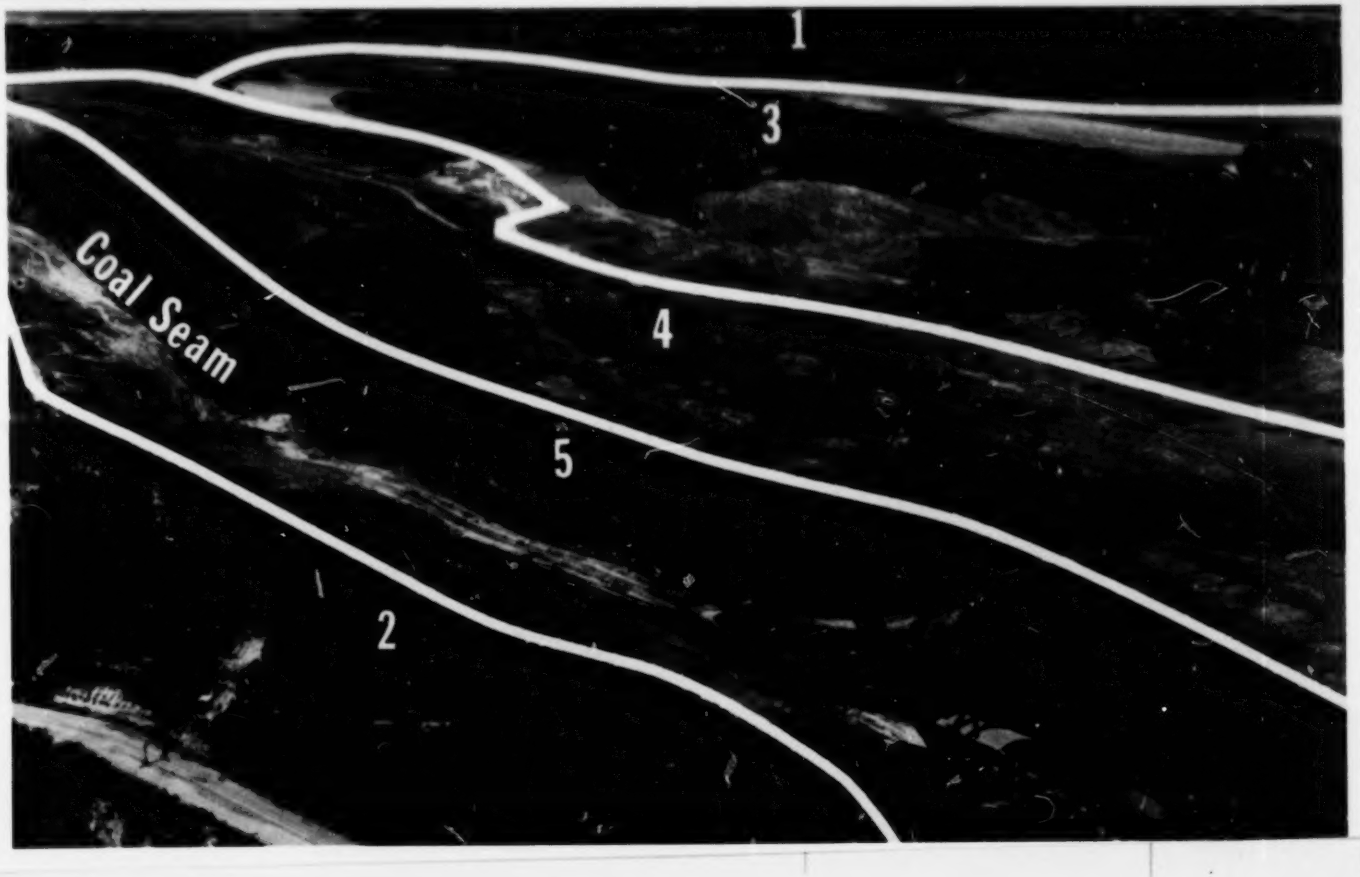

Figure 109. Various phases of surface mining and reclamation in a Kentucky coal mine: (1) virgin farm lands, (2) reforested area mined in 1947-1951, (3) reclaimed area mined 1954-1968, (4) reclaimed areas mined 1970-1971, partially contoured but not yet seeded, and (5) current mining operation with exposed coal seam.

most surface-mined land. Most research now underway in government and industry is in six categories: revegetation, chemistry of overburden and spoils, hydrology, earth movement and placement, haulroads, and land-use potentials.

RECLAMATION PLANNING

The specific planning problem with regard to reclamation is one of maximizing production and maintaining environmental quality to be consistent with the constraints imposed by the regulatory agencies, and the need to satisfy the profit objectives of the companies. Mining actions, on one hand, use some resources to produce coal, and at the other extreme, produce waste products, both of which affect the environment. There are direct costs and revenues to the company and public, as well as indirect costs and benefits which must be coordinated in the planning. The necessary planning steps are (1) to make an inventory of the premining conditions, (2) to evaluate and decide on the postmining requirements of the region to accomodate the needs and desires of the affected groups, (3) to analyze alternative mining and reclamation schemes to achieve the requirements optimally, and (4) to develop an acceptable integrated mining and reclamation scheme most suitable under the technical, social, and economic conditions.

For planning reclamation, information is needed in several areas. The initial concern from the environmental standpoint was the ability to reclaim the land in a minimal manner; today, however the environmental studies prior to launching a strip mine are much more detailed. The ecosystem factors that need to be considered today are set forth in Table 54. The listing is not complete, but points to the recognition of the fact that any attempt to extract mineral resources involves a change in the physical, biological, social, physiosocial, biosocial, and psychosocial factors. From the initial construction of access facilities to prospect to the final completion of the mining and beneficiation operations, the ecosystem balance is continually being altered. A discussion of the major factors as they affect the reclamation decision follows.

Physical and chemical attributes of the premining environment are factors in several areas of design and planning, particularly in the engineering design of excavation, materials handling, and ground support systems. They are also necessary in assessing water quality control of the operation and determining the potential future land uses.

Geological aspects of the coal occurrence (e.g., attitude, depth, thickness, stratigraphy, etc.) is the basis for establishing alternative mining and reclamation schemes including slope stability. The nature of the overburden, especially those components as sulfur, having potential environmental impacts must be studied. The structural and stratigraphic evaluation which effect the hydrological conditions are critical for effective mining and reclamation. Figure 110 and Table 55 show the type of analyses that were done in an exposed highwall in southern Ohio.

TABLE 54. INFORMATION NEEDS ON ECOSYSTEM FACTORS FOR SURFACE MINE PLANNING AND LAND USE

I. SOCIOECONOMIC INVENTORIES

(a) Land demands for mining and associated activities, and availability, encroachment into critical areas, other resources affected and foregone benefits.

(b) Past, current and future land use plans for the area-industry, agriculture, wilderness, etc.

(c) Transportation network

(d) Recreation networks

6-1/2"

(e) Archeological and historical interests

(f) Demographic and population characteristics
- (i) employment patterns
- (ii) tax base
- (iii) land value alterations and their impacts on local government
- (iv) attitudes of public, political, and educational institutions in the community/region

9-1/8"

II. GEOLOGIC ANALYSIS

(a) Topography

(b) Overburden Characteristics
- (i) stratigraphy
- (ii) physiography
- (iii) geomorphology
- (iv) chemical nature

(c) Soil and Toplayer Characteristics
- (i) physical
- (ii) chemical
- (iii) limitations for vegetation and future land uses
- (iv) limitations for engineering

III. COAL CHARACTERIZATION

(a) Coal seam attitude

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TABLE 54. (continued)

III. COAL CHARACTERIZATION (continued)

- (b) Coal seam thickness

- (c) Coal seam analyses
    - (i) proximate analyses
    - (ii) ultimate analyses
    - (iii) washability analyses

IV. TERRESTRIAL ECOLOGY

- (a) Natural vegetation, characterization, identification of survival needs

- (b) Crops

- (c) Game animals

- (d) Resident and migratory birds

- (e) Rare and endangered species

V. AQUATIC ECOLOGY

- (a) Aquatic animals - fish, waterbirds, resident and migratory

- (b) Aquatic plants

- (c) Characterization, use and survival needs of aquatic life system

VI. WATER RESOURCES

- (a) Surface hydrology
    - (i) watershed considerations
    - (ii) flood plain delineation
    - (iii) surface drainage patterns
    - (iv) amount and quality of surface run-offs

- (b) Ground water hydrology
    - (i) ground water table
    - (ii) aquifers
    - (iii) amount and quality of ground water flows
    - (iv) recharge potential

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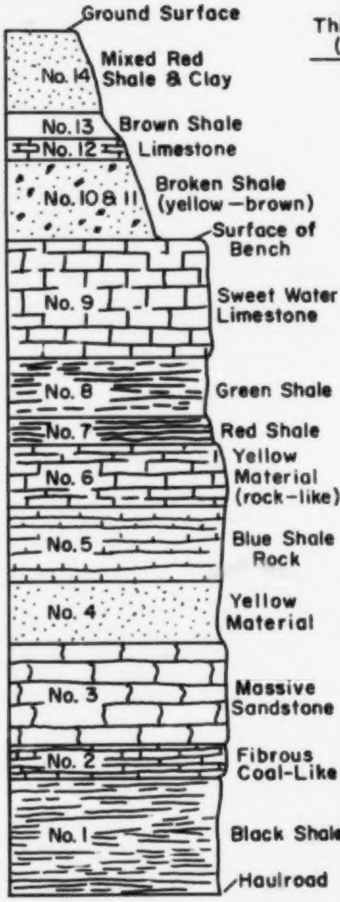
TABLE 54. (continued)

VII. CLIMATOLOGY
  - (a) Precipitation
  - (b) Wind - airflow patterns, turbulence
  - (c) Humidity
  - (d) Temperature
  - (e) Climate type
  - (f) Growing season

VIII. OTHER
  - (a) Noise
  - (b) Paticulates in the air
  - (c) Aesthetics

OHIO POWER HIGHWALL STUDY
East Pit Highwall


Ground Surface
No. 14
No. 13
No. 12
No. 10 & 11
Surface of Bench
No. 9
No. 8
No. 7
No. 6
No. 5
No. 4
No. 3
No. 2
No. 1
Haulroad

Layer	Thickness (feet)	pH	Fe (ferric)	Sulfates	Soluble Salts	Toxic Aluminum
			POUNDS PER ACRE			
No. 14 Mixed Red Shale & Clay	4	5.3	.4	40	320	.42
No. 13 Brown Shale	1.5	7.7	.2	22	420	0
No. 12 Limestone	1	8.1	.3	30	100	0
No. 10 & 11 Broken Shale (yellow—brown)	4	8.2	.1	25	360	0
No. 9 Sweet Water Limestone	6					
No. 8 Green Shale	3	8.1	.3	91	800	0
No. 7 Red Shale	1.5	7.8	.5	82	940	0
No. 6 Yellow Material (rock-like)	3	8.0	.5	61	520	0
No. 5 Blue Shale Rock	4	8.2	.3	61	660	0
No. 4 Yellow Material	3	3.8	22.5	106	3,200	.72
No. 3 Massive Sandstone	5	3.4	23.5	289	5,000	.66
No. 2 Fibrous Coal-Like	1.66	3.1	25.0	310	5,500	.54
No. 1 Black Shale	6	8.4	1.3	61	1,200	0

Source: Farm Clinic of U.S.

Figure 110. Analysis of the east pit highwall, a problem wall 5 years old, Ohio Power Highwall Study (Krause, 1973).

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TABLE 55. CHEMICAL ANALYSES OF THE EAST PIT HIGHWALL OHIO POWER HIGHWALL STUDY, 1967 (SOURCE: THE FARM CLINIC, WEST LAFAYETTE, IND.) (KRAUSE, 1973)

Sample No.	Organic Matter		Total Sol. Salts	Nitrogen		P_2O_5	K_2O	Ca	Mg	Ferric Fe	Ferrous Fe	Mn	B	Cu	Zn	SO_4	Toxic Al	Mo
				NO_3	NH_3													
	%	pH	lb/acre															ppb
4	0.9	3.8	3200	10.8	42	1.1	91	1365	270+	22.5	0.2	3.0	0.34	0.8	0.65	106	0.72	12
6	0.8	8.0	520	9.2	46	4.4	112	3500	270+	0.5	0.1	4.5	0.36	1.0	0.13	61	0	62
10	0.9	8.2	360	12.7	43	4.2	40	3450	270+	0.3	0.1	0.6	0.40	0.8	0.11	25	0	75
12	0.8	8.1	180	8.5	47	2.6	44	3640	270+	0.4	0.3	0.5	0.39	0.5	0.09	30	0	50
13	1.0	7.7	420	9.7	45	6.1	82	3250	270+	0.3	0.2	1.4	0.42	0.8	0.16	22	0	58
14	0.8	5.3	320	8.1	49	1.5	52	3080	114	0.5	0.4	4.5	0.37	2.1	0.50	40	0.42	25
Interpretation																		
Low	---	---	---	<20	<30	<5	<80	<500	<100	---	---	<1	<0.32	<1	<1	---	---	<25
Medium	---	---	---	20-40	30-50	5-20	80-150	500 to 1000	100-180	---	---	1-4	0.32 to 0.37	1-4	1-4	---	---	25-50
Satisfactory	---	---	---	>40	>50	>20	>150	>1000	>180	---	---	>4	>0.38	>4	>4	---	---	>50
Excessive or toxic	---	---	---	---	---	---	---	---	---	---	---	>20	>0.45	--	--	---	>1	--

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Soils must be considered to ensure their proper removal, placement, and postmining disposition. While the definition and identification of the material to be classified as topsoil is fraught with difficulties, the characteristics of the material to serve as soil after mining must be known. The characteristics of topsoil should be such that it should ensure an effective growth of vegetative cover than other materials. Topsoil and subsoil, though not always the most ideal materials, do have certain advantages over other materials, including higher organic matter, better moisture-holding capacity, natural seed source, and soil microorganisms. The removal and storage of topsoil and replacement of it on top of the graded soil, in many instances leads to better revegetation potential, which in turn leads to better erosion control. In many older mining districts, spoil banks are only sparsely revegetated. This is due in part to the lack of nutrients in the spoil and the water flow characteristics of the restored strata. In cases where bedrock material, shale, sandstone, siltstone, and fireclay have been mixed with the original topsoil covering, inadequate fine-grained matrix material and organic matter may be available to provide nutrients or moisture-holding capacity. Too frequently, the fine grained material sinks through the interstices in the coarse matrix. High infiltration capacity, high porosity and high permeability are too commonly encountered when no attempt is made at toplayer restoration and compaction. Color of the spoil has an effect on the surface temperatures of the spoil banks. Temperatures in excess of 150°F have been reported on spoil surface composed of dark shales.

Regrading of spoils alone may be inadequate to provide a growth medium. In fact, the mining operation provides an opportunity to modify the soil strata to include burial of toxic soil strata and complete rebuilding of soil layers at the top. A few of the soil modification and reconditioning methods include treatment with lime or limestone, fertilizers, and other additions such as mulch, fly ash or sewage sludge to adjust pH, nutrient and physical conditions. Two important factors to be determined here are the amount and depth of application of these modifiers.

Topography influences the selection of the type of mining, and mining equipment. These topics were discussed in an earlier section. Severe terrains also influence the methods of revegetation, and limit alternatives for future land use.

Extremes in climate affect the selection of equipment. Seasonal fluctuations determine desirable planting periods. In addition to the problem of permafrost in severe cold climates, operation of mobile equipment becomes difficult. Heavy rains can cause erosion complicating water quality control, sediment formation, and the establishment of vegetation.

Hydrology, both surface and underground, together with the physical and chemical characteristics of overburden and coal, is important for any reclamation program planning. A knowledge of water movement and quality in the mine area and modification during mining is essential. In all cases, erosion and sediment production are immediate concerns in the critical postmining, pre-revegetation time period, although their control during storms is a long-term problem that requires detailed planning. The Surface Mining

Control and Reclamation Act of 1977 requires that the surface mining operation be so conducted as to minimize the disturbance (1) to the prevailing hydrologic balance at the mine site and in associated offsite areas, and (2) to the quality and quantity of water in surface and ground water systems both during and after surface coal mining operations. Among the specific provisions of the law for the avoidance of acid or toxic mine drainage, included are such measures as:

1. preventing or removing water from contact with toxic producing deposits.
2. treating drainage to reduce toxic content which adversely affects downstream water upon being released to water courses.
3. casing, sealing, or otherwise managing boreholes, shafts, and wells to keep acid or other toxic drainage from entering ground and surface waters.

With regard to hydrologic balance, other provisions of the Act deal with factors such as suspended solids and recharge capacity of the mined area. The negative impacts of surface mining are to be minimized by:

1. conducting surface coal mining operations so as to prevent, to the extent possible using the best technology currently available, additional contributions of suspended solids to streamflow or runoff outside the permit area. In no event shall contributions be in excess of requirements set by applicable State or Federal law.
2. constructing any siltation structures, prior to commencement of surface coal mining operations. Such structures are to be certified by a qualified registered engineer and constructed as designed and as approved in the reclamation plan.
3. cleaning out and removing temporary or large settling ponds or other siltation structures from drainways after disturbed areas are revegetated and stabilized; and depositing the silt and debris at a site and in a manner approved by the regulatory authority.
4. restoring recharge capacity of the mined area to approximate premining conditions.
5. avoiding channel deepening or enlargement in operations requiring the discharge of water from mines.
6. preserving throughout the mining and reclamation process the essential hydrologic functions of alluvial valley floors in the arid and semiarid areas of the country.
7. other such actions as the regulatory authority may prescribe.

Other volumes of the final report (Volumes IV and V) have dealt with the subject of water quality and quantity in greater detail.

While the scope of the present study does not include a detailed discussion of the biological characterization of the mining area, such typification is important in the selection of a reclamation plan. In many

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cases, the law provides for the protection of unique or endangered species of plants and animals which require specific consideration. Such provisions may even prevent the mining of coal.

Sociohumanistic factors, though very important, are another study area which is outside the scope of the present report. Nonetheless, a reclamation plan must consider attitudes of the public and current and future uses of the land. Aesthetic or historic value of the land prior to mining may also be important enough to preclude mining.

LAND-USE ANALYSIS

The current land-use patterns may provide the best guideline for reclamation. For example, if there is a need for pasture lands, forested hillside land adjoining a pasture may be reclaimed for pasture after mining. Similarly, orphaned mined land in a wilderness area may be more worthwhile if restored as a forest. Future land-use plans contrary to premining use are unusual and not the common practice. Acceptance of mountaintop removal and valley fill mining methods which create new flat areas in regions where such areas are in short supply reflect the permanent, positive potentials of the temporary, drastic disturbance of lands due to mining. Identification of the current and probable future land uses can be summarized into the following five categories:

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1. "Wilderness" or unimproved use
2. Limited agriculture or recreation, with little development, such as grazing, hunting or fishing or as timber land
3. Developed agriculture or recreation, such as crop land, water sports, vacation resorts
4. Suburban housing or light commercial and industry
5. Urban housing or heavy commercial and industry

Analysis of alternative long-term uses requires compliance with socioeconomic and public constraints as well as the economic goals of the mining company. Discussion on this subject is necessarily limited here since another report on land-use planning has dealt with this subject in greater detail. Irrespective of the choice of short-term and long-term land use, these plans must be linked to the mining plan in space, time and method. Definition of the long-range plans do achieve a purpose by minimizing disturbance unnecessary for the determined future use and delimiting those reclamation activities that are not a part of the normal mining or land-use plans. Reference has already been made to the enactment of legislation at the federal and state levels pertaining to surface mining and reclamation.

Several provisions of the Surface Mining Control and Reclamation Act of 1977 and the interim regulations deal with land-use planning. Specifically, with regard to obtaining a mining permit, a reclamation plan must be submitted which shall be of sufficient detail to demonstrate that reclamation can be achieved as required by Federal or State programs. Such a plan must cover, among other items, the following important points with regard to the uses of land:

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1. the uses existing at the time of the application, and if the land has a history of previous mining, the uses which preceded any mining.
2. the capability of the land prior to any mining to support a variety of uses giving consideration to soil and foundation characteristics, topography, and vegetative cover.
3. the use which is proposed to be made of the land following reclamation, including a discussion of the utility and capacity of the reclaimed land to support a variety of alternative uses and the relationship of such use to existing land use policies and plans, and the comments of any owner of the surface, State, and local governments or agencies thereof which would have to initiate, implement, approve, or authorize the proposed use of the land following reclamation.
4. a detailed description of how the proposed post-mining land use is to be achieved and the necessary support activities which may be needed to achieve the proposed land use.
5. the consideration which has been given to making the surface mining and reclamation operations consistent with surface owner plans and applicable State and local land-use plans and programs.
6. the consideration which has been given to developing the reclamation plan in a manner consistent with local physical environmental and climatological conditions.

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Even otherwise, mining companies today have to meet with nongovernmental, historical, wildlife, and other environmental groups and allay their fears. The integrated mining, reclamation and land-use planning diagram (Figure 111) sets out an iterative procedure for the evaluation of mining and reclamation plans. The feedback loops in the diagram serve to indicate the procedure which, when the plans are optimal will lead to the mining and reclamation of the coal seam. Shown in the Figure is also another set of interactions that takes place before a mining plan is put into practice. The iterative loop through which the mining and reclamation plans have to pass includes not only the federal, state and local agencies but the special interest groups and the general public. Finally, the ecosystem factors to be considered in the development of a mine today are varied, many, and increasing. The planning of surface mines, thus, is very involved and thorough, and the objective of planning is the long-term use of the land as a prime natural resource.

ECONOMIC FEASIBILITY STUDIES

In a free enterprise society, it is difficult to continue in business unless there is an adequate rate of return in monetary terms on the investment. The goals of environmental protection and mineral resource extraction must therefore be analyzed in terms of both the economic return and the constraints and incentives of the public and the government. At the mine level, it is necessary to establish the direct cash cost of reclamation. The consideration of internal, direct costs, and income from mineral sales, by more easily quantifiable rate of return analysis, is traditional.

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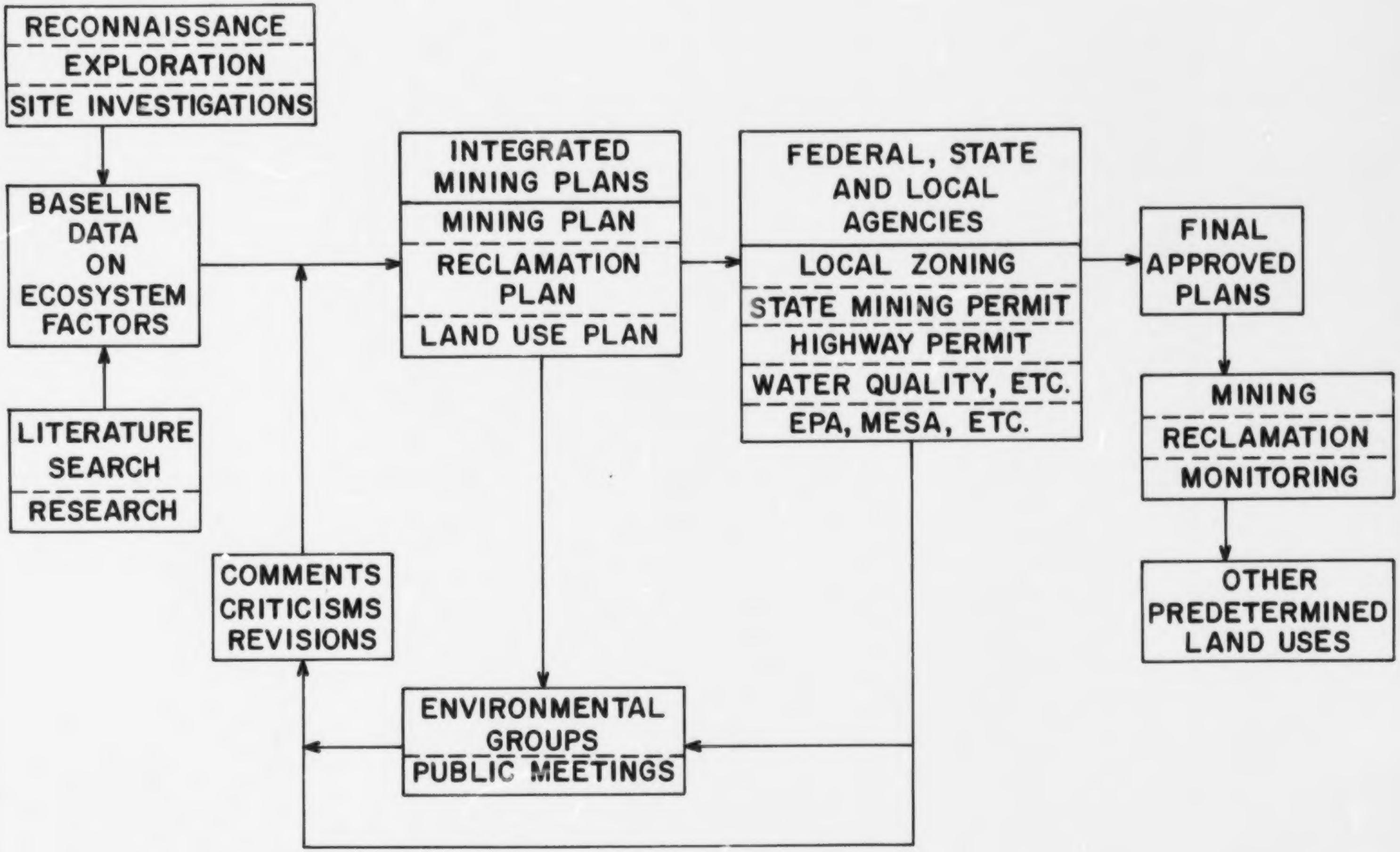
RECONNAISSANCE
EXPLORATION
SITE INVESTIGATIONS
BASELINE DATA ON ECOSYSTEM FACTORS
LITERATURE SEARCH
RESEARCH
COMMENTS CRITICISMS REVISIONS
INTEGRATED MINING PLANS
MINING PLAN
RECLAMATION PLAN
LAND USE PLAN
ENVIRONMENTAL GROUPS
PUBLIC MEETINGS
FEDERAL, STATE AND LOCAL AGENCIES
LOCAL ZONING
STATE MINING PERMIT
HIGHWAY PERMIT
WATER QUALITY, ETC.
EPA, MESA, ETC.
FINAL APPROVED PLANS
MINING
RECLAMATION
MONITORING
OTHER PREDETERMINED LAND USES

Figure 111. Integrated and interactive surface mining, reclamation and land-use planning.

Grim and Hill (1974) have summarized the results of several studies on reclamation costs. Mathematica Incorporated (1973) has estimated that reclamation costs will be about 8% of the total mining cost (Table 56). Other cost figures in literature vary from $1,236 to $19,768 per hectares ($500 to $8,000 per acre) depending on the severity of the land effects and the reclamation requirements. Doyle et al. (1974), have prepared a useful report on the analysis of pollution control costs. Their report addresses the costs involved in revegetation, acid mine drainage, stream diversion, strip mine spoil and waste, backfilling and regrading, etc. Table 57 is extracted from that report to indicate the various factors that affect backfilling and grading costs.

Since reclamation costs are site specific, the requirements of labor, equipment and supplies must be estimated. A convenient basis for estimation purposes may be to identify the acreage disturbed for the recovery of coal from underneath an acre of ground. The costs of reclamation are calculated for the disturbed land and then assigned to the tons of coal recovered. Reclamation costs are preferably indicated both in $/acre and $/ton.

To minimize environmental disturbances and to maximize land use, it is necessary to include the sociohumanistic benefits and costs. These are not as easily quantified, defy traditional engineering approaches, and raise conflicts between the various organizations' objectives. The consideration of these benefits and costs are outside the scope of this report. The future of surface mining hinges partially upon the ability to reclaim land in a satisfactory manner. Where the environmental impacts cannot be clearly ascertained, it may be that the mineral resource should not be mined. However, in a majority of the instances, knowledge and technology are available to achieve acceptable impacts. Results of several studies reveal that no major breakthroughs are indicated in the near future. Only through concerted efforts will evolutionary processes result that adapt present equipment with mining cycles which reflect in greater social concern for environment.

TECHNICAL FEASIBILITY STUDIES

Technical feasibility is a prime requisite for the adoption of any reclamation plan. It is preferable to key-in the reclamation plans with mining plans since nonconcurrent reclamation methods tend to be environmentally less optimal and more expensive. Reclamation efforts initiated long after the mining activity has ceased generally result in reclamation that serves a limited land use. Concurrent reclamation with positive controls aids in the development of well-conceived future land uses with a long-term program to finish and maintain the reclamation plans. Concurrent methods allow total spoil manipulation and/or complete spoil profile construction.

Technical requirements of reclamation will vary depending upon the degree of restoration. In the standard surface mining approaches of today, selective replacement of material is relatively simple. Contour mining approaches such as block cut, box cut, haul back, mountaintop removal and modified area mining are generally flexible to allow segregation and

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TABLE 56. ESTIMATED AVERAGE PRODUCTION COSTS @ AVERAGE STRIPPING RATIO = 8:1 and 0.5 ACRES DISTURBED PER THOUSAND TONS OF COAL PRODUCED* (MATHEMATICA, 1973)

COST ELEMENT	COST ($/TON)	COST (% OF TOTAL)
Sediment structure	0.12(1)	2.9
Acreage fees	0.01(1)	0.2
Bonding	0.00	0.0
Scalping	0.08(1)	1.9
Stripping	2.40	57.5
Overburden haulage	0.05	1.2
Coal loading	0.10	2.4
Coal haulage	0.50	12.0
Backfilling and grading	0.08(1)	1.9
Revegetation	0.03(1)	0.7
Royalties	0.50	12.0
Severence tax	0.30	7.3
Total	$4.17	100.0

* Acre = 0.40 hectares; short tons = 0.907 metric tons

(1) Reclamation costs. These costs are equivalent to $0.32 per ton or 7.6% of the total costs

Note The primary assumption underlying these estimates is that the stripping ratio is 8:1; the cost balance would change somewhat if the assumed stripping ratio were to change.

TABLE 57. STRIP MINE RECLAMATION PROJECT VARIABLES AFFECTING BACKFILLING AND GRADING COSTS (DOYLE, 1974)

1. Geographic location

2. Topographic setting (original, pre-reclamation and final ground slopes)

3. Type of strip mine:
   a) area, b) contour, c) area-contour, d) other

4. Coal seams mined and thickness

5. Inclination of coal seams in back of highwall:
   a) dip, b) rise, c) horizontal

6. Condition of coal seams in back of highwall:
   a) not mined, b) auger mined, c) drift mined (entries opened or caved, d) mine workings exposed by stripping operation

7. The probable hydraulic head that could develop if coal in back of highwall was mined

8. Strip mine area information:

   A. Length, width, and area covered by spoil before reclamation
   B. Highwall height (maximum and average height)
   C. Highwall length
   D. Number of cuts
   E. Total area affected during reclamation in acres (including area above highwall and outside of slopes)
   F. Volume of spoil to be moved
   G. Average haul distance for backfilling and grading
   H. Texture of spoil
   I. Amount of large rock and material requiring special handling of mining timbers, machinery, and debris, junked cars, and other solid waste
   J. Amount and reactivity of pyritic material (minerology and mode of occurrence. For example finely dispersed; single crystals or crystal aggregates; coatings on joint surfaces; in form of lenses, layers or modules; "sulfur balls"; pyritic shales, etc.)
   K. Clearing and grubbing requirements

9. Type of backfill:
   a) contour, b) pasture-reverse slope, c) swallowtail, d) head of hollow, 3) submergence, f) other

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TABLE 57. (continued)

10. Physical sealants for covering toxic material
a) none, b) clay, c) bituminous material, d) plastic material, e) other

11. Compaction desired:
a) none, b) only toxic materials, c) all spoil material with exception of upper layer up to 0.91 meter (1 to 3 feet)

12. Accessibility factors:

A. Right-of-way problems
B. Ingress and egress construction (include clearing and grubbing for access and post-construction revegetation)
C. Other factors affecting access

13. Surface and subsurface ownership of strip-mined area. Also, ownership of properties for ingress and egress:
a) public, b) private, c) in process of being acquired or line placed on property, d) abandoned, 3) temporary easement, f) other

14. Time of year reclamation performed

15. Weather conditions during reclamation period(s)

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selective replacement. Two such plans are illustrated in Figures 112 and 113 for contour mining. Larger scale area methods employing traditional overcasting are less flexible. It is common practice to handle overburden by direct overcasting preferably with no rehandle. The problem with simple overcasting with a large machine is that the spoil can be cast only at a limited distance from the cut, and then only at a limited range of angles to the line of travel of the equipment. However, selective placement of spoils in these cases can also be achieved by a combination of equipment and methods. For example, the requirement to save topsoil and to spread it over the regraded land has led the mining companies to the use of scrapers that remove the topsoil, run around the pit and dump over formerly graded spoil. Selection of shovel-truck systems, as opposed to that of conventional overcasting, also provides the needed flexibility in soil placement. In fact, the overburden conditions can be closely monitored, and the placement of the overburden so achieved to ensure 1) burial of toxic spoils and less desirable material such as boulders, or impermeable clays, 2) topsoil replacement, 3) desired stratigraphic sequence in the spoil, and 4) flatter or other desired valley configurations. Additionally, truck shovel systems permit blending in the pit for coal quality and greater recovery of coal. The goal for premining planning and scheduling for resource extraction and reclamation is to develop such steps as an integral part of the mining method.

The question of restoration versus adaption of contour is one which is decided by the general mining approach. In strict contour mining, in which the seam is followed around a hill, the restoration to approximate original contour with excess spoil properly disposed in the valleys is generally most appropriate. Where there are flat lands, and the need is for small hills, terrace and shallow hollows, the indicated approach may be to restore some degree of relief by leaving the mined-out areas and spoils in such a reclaimed state that the future potential may be for grazing, wildlife, or recreation. In areas where the lands are used for agriculture prior to mining, the plan may be to re-establish all mined areas to a more tillable topography.

In backfilling, three major areas of long-term concern are 1) slope stability, 2) ground water control, and 3) water quality maintenance. The primary source of slope failure is the prolonged action of water, through erosion, and mass wasting. The primary result of water's actions on slopes is erosion and sediment production. Long uninterrupted slopes are undesirable as compared to ones with terraces and diversions which decrease slope lengths and direct runoff water to safe outlets. Additionally, while backfilling, the permeability of the fill for water perculation can be modified. Such modifications are achieved by sealing or leaving the coal seam open and by selective placement of the spoil. One of the concerns of water quality control, in addition to the acid potential, is the sediment. The interconnection between slope stability and water quality (sediment) is enhanced by apparently conflicting approaches to these problems. Gross stability is enhanced by establishing grade and cover to minimize water infiltration and permeation. Conversely, erosion and sediment control are achieved by encouraging infiltration and permeation to minimize runoff.

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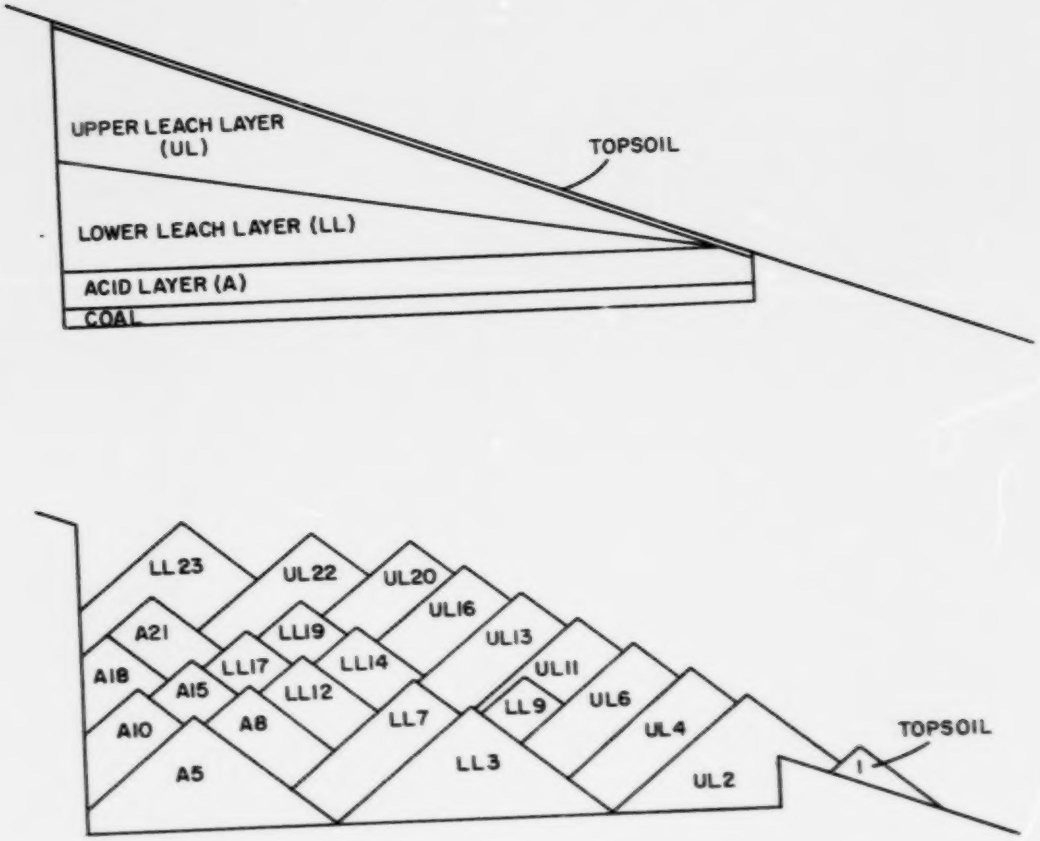
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TOPSOIL
LOWER LEACH LAYER (LL)
ACID LAYER (A)
COAL
LL23
UL22
UL20
UL16
A21
LL19
UL13
A18
LL17
LL14
UL11
A15
LL12
LL9
UL6
A10
A8
LL7
UL4
TOPSOIL
A5
LL3
UL2
1

Figure 112. Movement of spoil within a block to insure segregation and burial of acid-bearing material (Saperstein and Secor, 1973).

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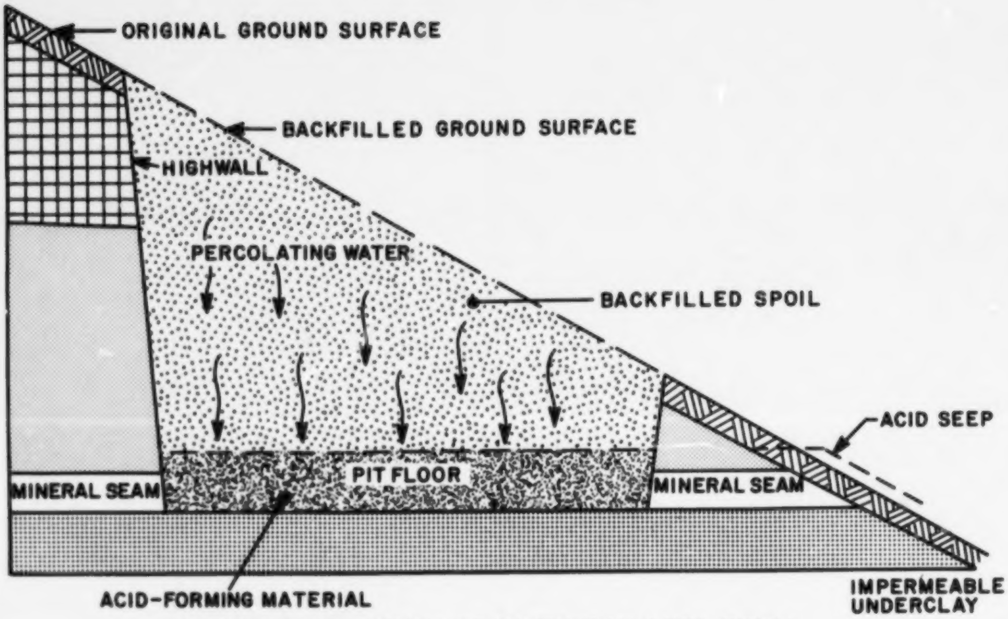
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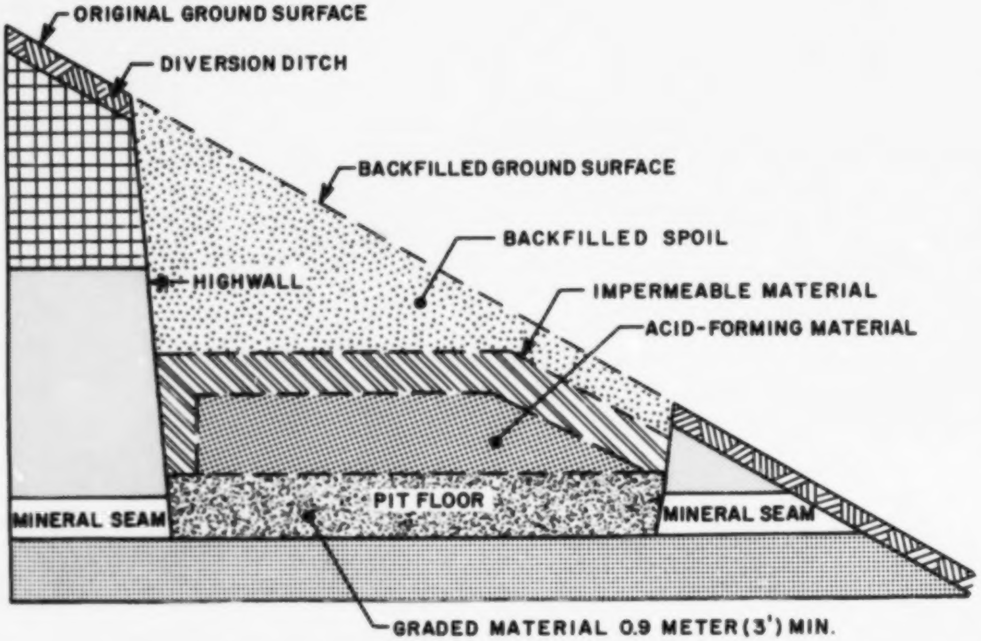
BACKFILLED GROUND SURFACE
HIGHWALL
PERCOLATING WATER
BACKFILLED SPOIL
ACID SEEP
PIT FLOOR
MINERAL SEAM
MINERAL SEAM
ACID-FORMING MATERIAL
IMPERMEABLE UNDERCLAY

POOR PLACEMENT RESULTING IN ACID SEEP.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ACID-FORMING MATERIAL
PIT FLOOR
MINERAL SEAM
MINERAL SEAM
GRADED MATERIAL 0.9 METER (3') MIN.

PROPER PLACEMENT FOR ACID FORMING MATERIAL BURIAL

Figure 113. Spoil placement for better environmental impact (Hill and Grim, 1975).

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The proper approach would entail, first, the establishment of as heavy a cover crop after backfilling as possible, to encourage vegetative interception and evapotranspiration as opposed to surface or subsurface runoff. However, not all of the water can be handled in this manner. Provision must be made for storms by designing clean-running interceptions and diversions for excessive rainfall.

Stability and water quality are inherent in the design of fills and backfills. If a suitable, large-rock drainage base and contact zone is established at the base contact of the fill, slippage can be minimized while rapid drainage of groundwater is enhanced. Such approaches to water quality and the control of stability are exemplified by the head-of-hollow or valley-fill concept, explained in the earlier section on mining methods.

Replacement of topsoil is practiced in mining where required by regulations. The dictum of topsoil replacement could be viewed as a requirement to replace or establish a layer of material of specific thickness, having certain physical and chemical characteristics, and primarily, capable of supporting some specific vegetation. In pre-mining planning, attention should be directed at identifying all strata potentially able to adapt as a surface covering. If topsoil is not available, it becomes necessary to consider soil amendments to establish vegetation. Common approaches to soil amendments involve the addition of lime, mulch, and fertilizer. Many mulches have been used to assist in establishing vegetative cover. Straw or hay can be manually applied, but they, as well as wood chips, are often spread mechanically or hydro-pneumatically. Mulch should be tagged down with tar, etc. Rapid growing annual plants and small grains can be used to establish ground cover, to leave organic matter in the soil, and to serve as a mulch for perennial establishment. The common practice today is to establish such quick-cover mulch crops prior to topsoiling to assure vegetative stabilization and to enhance the organic content and water-holding capability of the surface layer. Specification of the graded surface, furrowing or harrowing, is also done to act as seed and water traps. Care must be taken in selecting species to avoid competition.

Fly ash, the residue of coal-fired boilers, can be applied to increase nutrients, moisture and alkalinity and to ameliorate toxic materials present. The trace elements contained in fly ash are often important micronutrients, while fly ash is also rich in the common macronutrients except nitrogen. Nitrogen deficiency can be handled by planting inoculated clover, alfalfa, vetch, or other legumes. The use of fly ash may be particularly attractive when the power plant is located at or near the mine and the ash must be disposed of in some fashion (Adams, 1971).

Stabilized sewage sludge is another recyclable amendment to reclaimed-mined land. Sludge is rich in nutrients and may reduce acidity, as well as provide stable organic matter to form a humus. A related type of amendment is composting with municipal waste, which has been practiced in other countries but has seen limited application in the United States. Care must be taken, however, to avoid poisoning the soil with too high a level of heavy metals and salt from the sludge (Grim and Hill, 1974).

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In surface coal mining, positive control of environmental impacts is achieved by proper mining and reclamation techniques which include concurrent reclamation, overburden segregation, toxic material burial, and topsoiling. Grading and backfilling should immediately follow mining as the freshly spoiled material is easier to move, backfill, and grade. Revegetation should follow grading as soon as possible. The primary purpose of vegetation is to stabilize the spoil to prevent erosion and to ensure water quality and quantity control. In most cases, therefore, revegetation is best accomplished by the establishment of grasses and legumes, as soon after mining as possible, though the long range plan may call for reforestation or other land uses. It is possible to develop a time sequence for the various reclamation activities. Table 58 presents such a sequence. For operational management, this sequence must have specific dates for a particular operation.

MONITORING AND MAINTENANCE

Even with proper reclamation, rededication of the land for productive purposes may require more time. Grazing, for example, should not be allowed immediately on a new cover crop. In any case, use of the land is not definitely accomplished at the end of mining. A careful plan of monitoring and control, and maintenance and development are required.

Monitoring slopes to assure stability and control to maintain water quality are two important aspects. To minimize these problems initially and to prevent them from occurring in the future, at the time of grading it is necessary to keep all surfaces as minimally sloped as possible and under the angle of repose. Additionally, through regular inspections any failure or runoff is corrected by refilling, compacting, and rapid establishment of cover. To control the erosion affects of running water, it is necessary to place interceptor and diversion channels to reduce downslope flows.

Where the coal mine drainage phenomenon cannot be prevented, a long-term program to neutralize the discharge is necessary. In most surface-mining situations, much of the necessary layout is present in sediment-control ponds. Addition of limestone, lime, anhydrous ammonia (where no discharge to surface drainage is encountered), soda ash, sodium hydroxide, or other suitable neutralizing reagents can minimize acid formation. However, isolation of toxic material and concurrent backfilling is probably the most desirable approach to minimization of acid formation problems.

One other aspect of maintenance is the sediment and erosion control installations. With appropriate planning and no unusual storms, such maintenance of these features may be minimal. However, control structures may become clogged or filled and should be cleared and cleaned through regular inspections.

TABLE 58. TIME SEQUENCE FOR RECLAMATION ACTIVITIES

I. DURING SITE PREPARATION:

1. Install control measures (diversion, sediment traps and basins, etc.).
2. Clear and grub, market lumber if possible; stockpile brush for use as filters; run brush through woodchipper and use chips for mulch.
3. Stabilize areas around temporary facilities such as maintenance yard, power station, and supply area.

II. DURING OVERBURDEN REMOVAL:

1. Divert water away from and around active mining areas.
2. Remove topsoil and store it if possible and/or necessary (Figure 114).
3. Selectively mine and place overburden strata if possible and/or necessary.

III. DURING COAL REMOVAL:

1. Remove all coal insofar as possible.
2. Break or prevent damage to the strata immediately below the coal seam as desired.

IV. IMMEDIATELY AFTER COAL REMOVAL:

1. Seal the high wall if necessary.
2. Seal the low wall if necessary.
3. Backfill--bury toxic materials and boulders, dispose of waste, ensure compaction.

V. SHORTLY AFTER COAL REMOVAL:

1. Rough grade and contour, taking these factors into consideration (Figures 115, 116 and 117):
    - a. Time of grading, specific time limit; tied to advance of mining; seasonal considerations.
    - b. Slope steepness.
    - c. Length of uninterrupted slope.
    - d. Compaction.
    - e. Reconstruction of underground and surface drainage patterns.

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TABLE 58. (continued)

V. SHORTLY AFTER COAL REMOVAL:

2. If necessary, make mine spoil amendments (root zone), taking these factors into consideration:
   a. Type of amendment, fertilizers, limestone, fly ash, sewage sludge, or others.
   b. Depth of application.
   c. Top layer considerations--temperature (color), water retention (size consist, organics), mulching and tacking.

VI. IMMEDIATELY PRIOR TO FIRST PLANTING SEASON:

1. Fine-grade and spread topsoil, taking seasonal fluctuations into consideration (Figure 118).
2. If necessary, manipulate the soil mechanically by furrowing, deep-chiseling or harrowing, or constructing dozer basins.
3. Mulch and tack.

VII. DURING THE FIRST PLANTING SEASON:

Seed and revegetate, considering--time and methods of seeding; choice of grasses and legumes (Figures 119 and 120).

VIII. AT REGULAR, FREQUENT INTERVALS:

Monitor and control: slope stability; water quality, both chemical (ph, etc.) and physical (sediment); vegetation growth.

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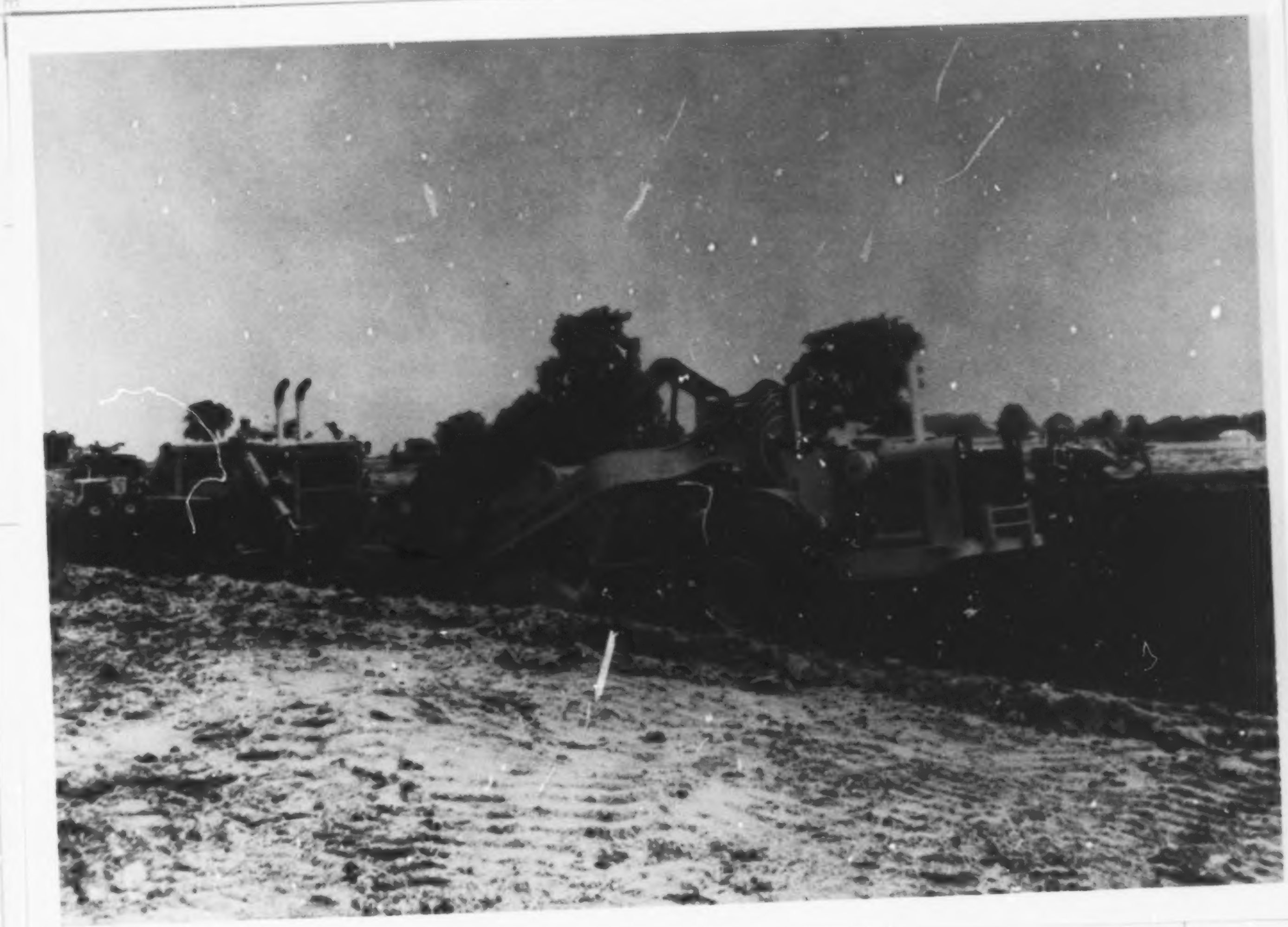

Figure 114. A pusher-scraper combination for topsoil and overburden removal.

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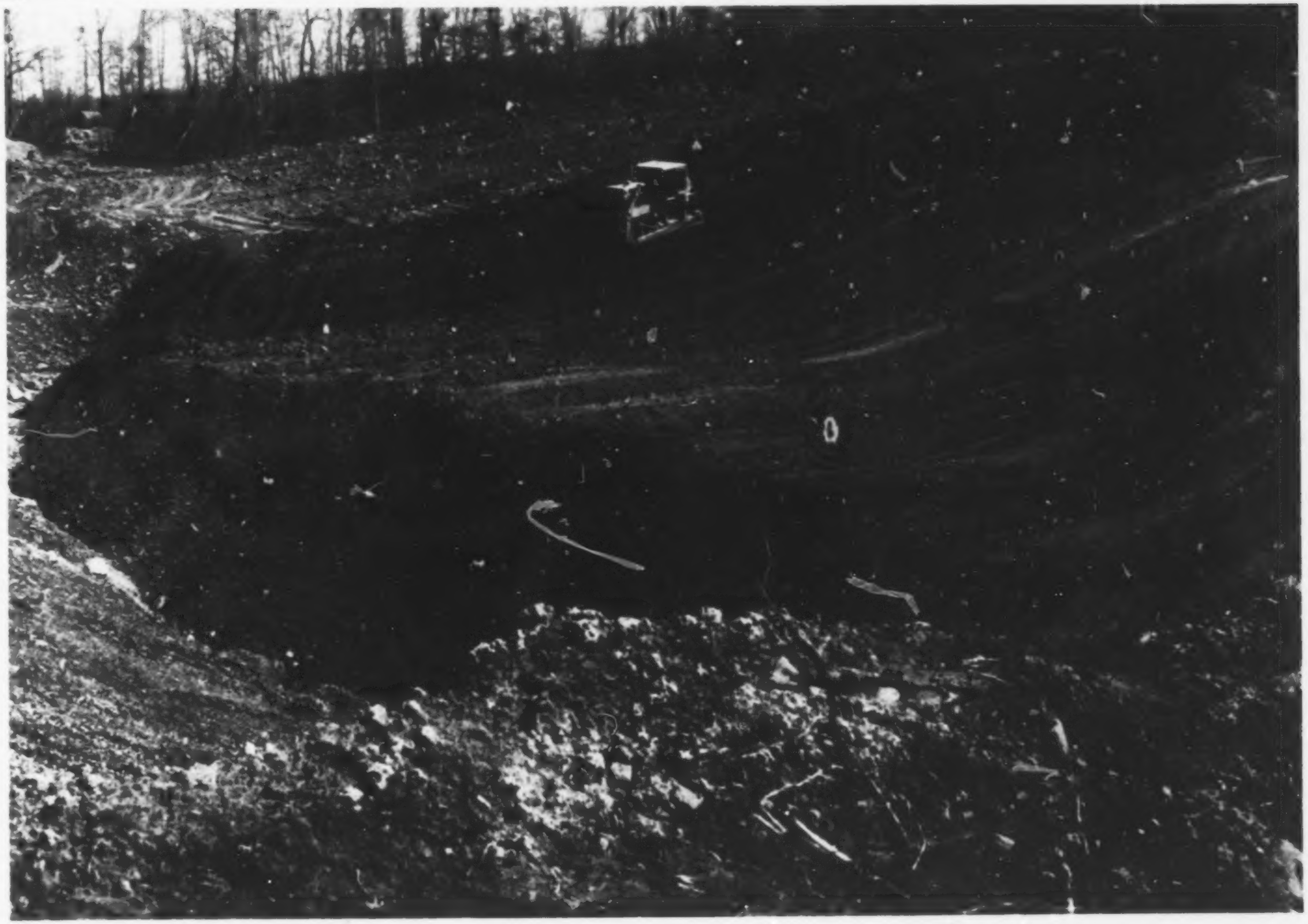

Figure 115. Large crawler tractor in the process of rough grading coal mine spoil banks in hilly terrain.

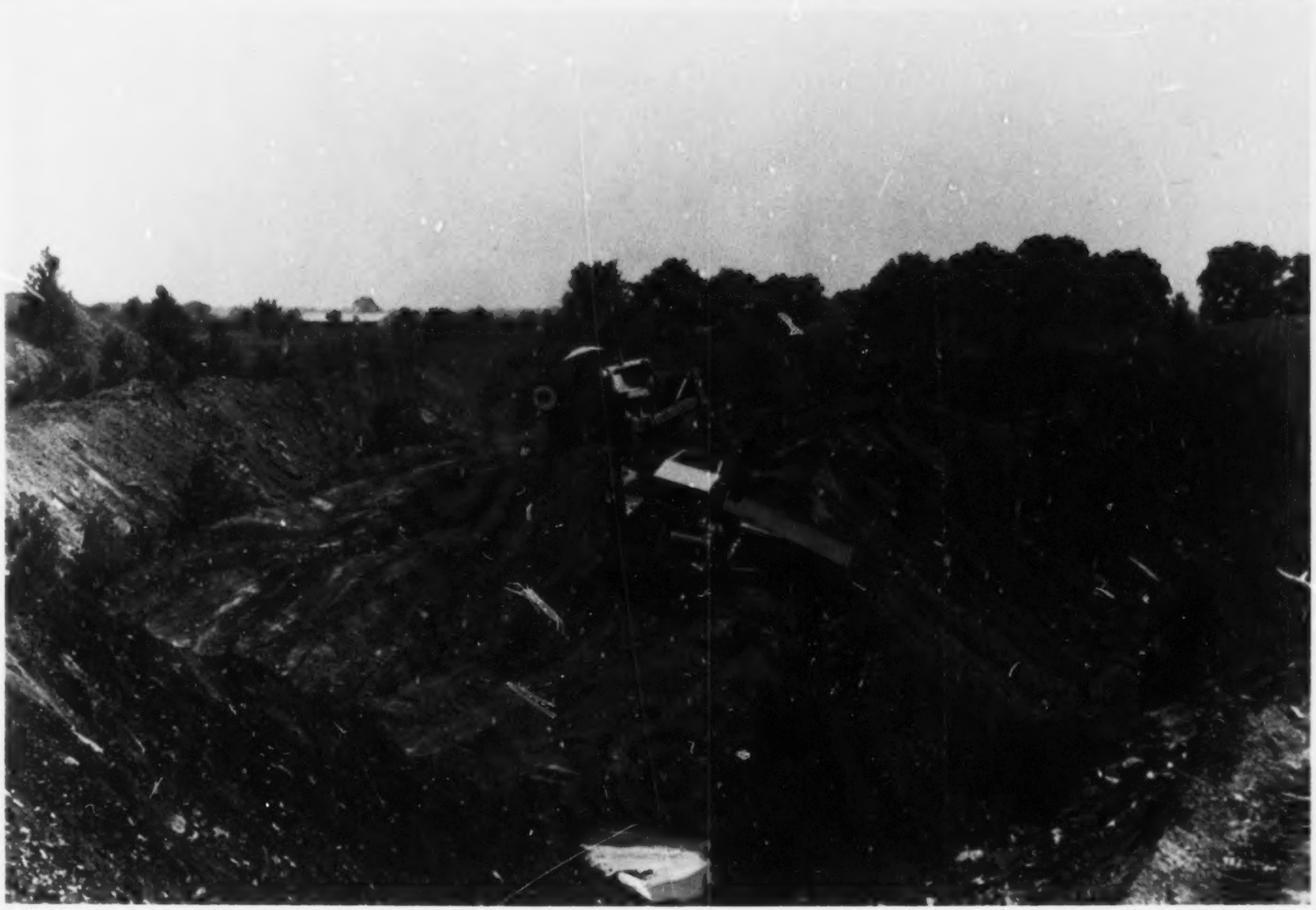

Figure 116. A giant bulldozer with an experimental 40-foot blade levels mined land in southeastern Kansas.

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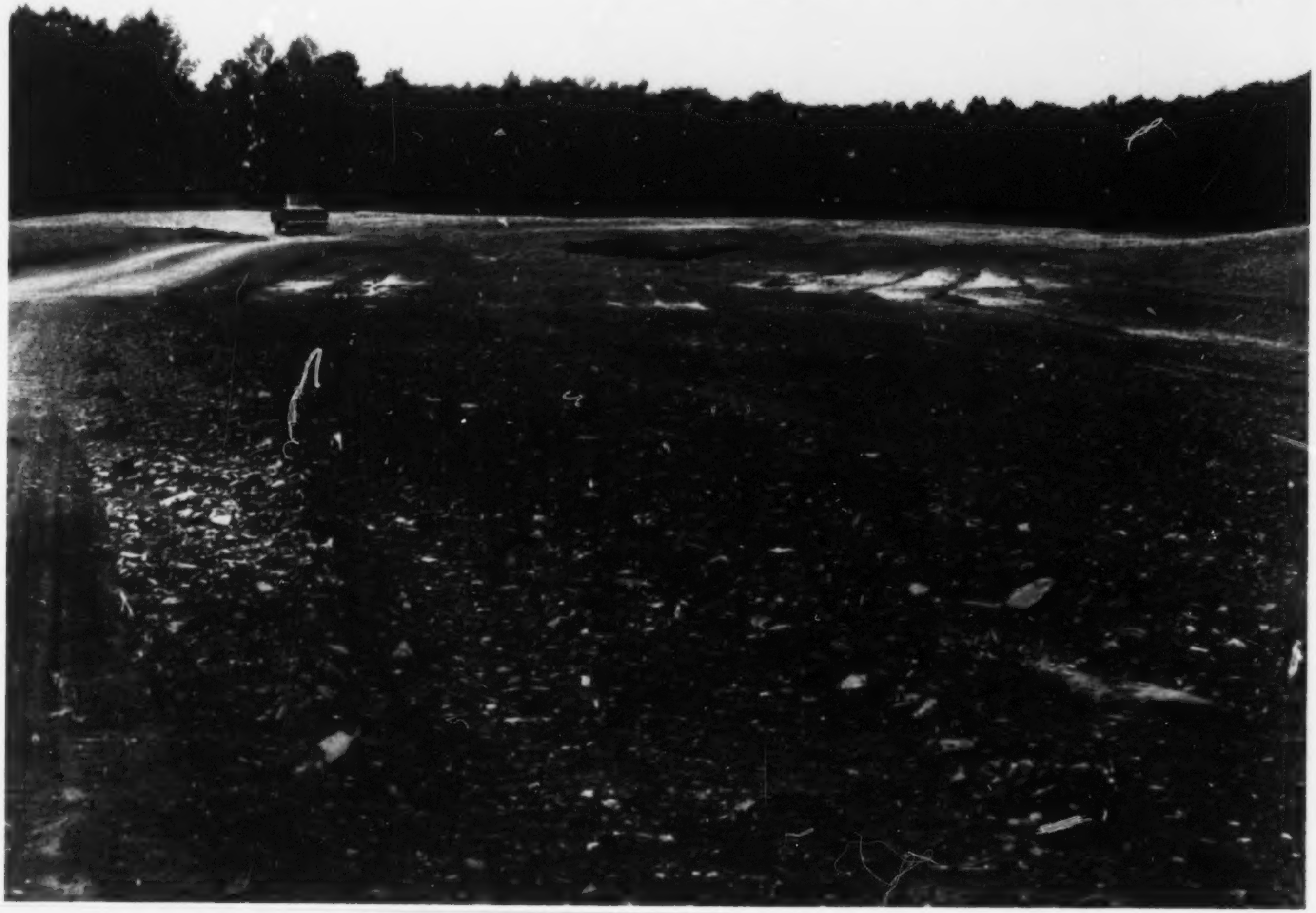

Figure 117. Completed rough grading of coal mine spoil banks in hilly terrain.

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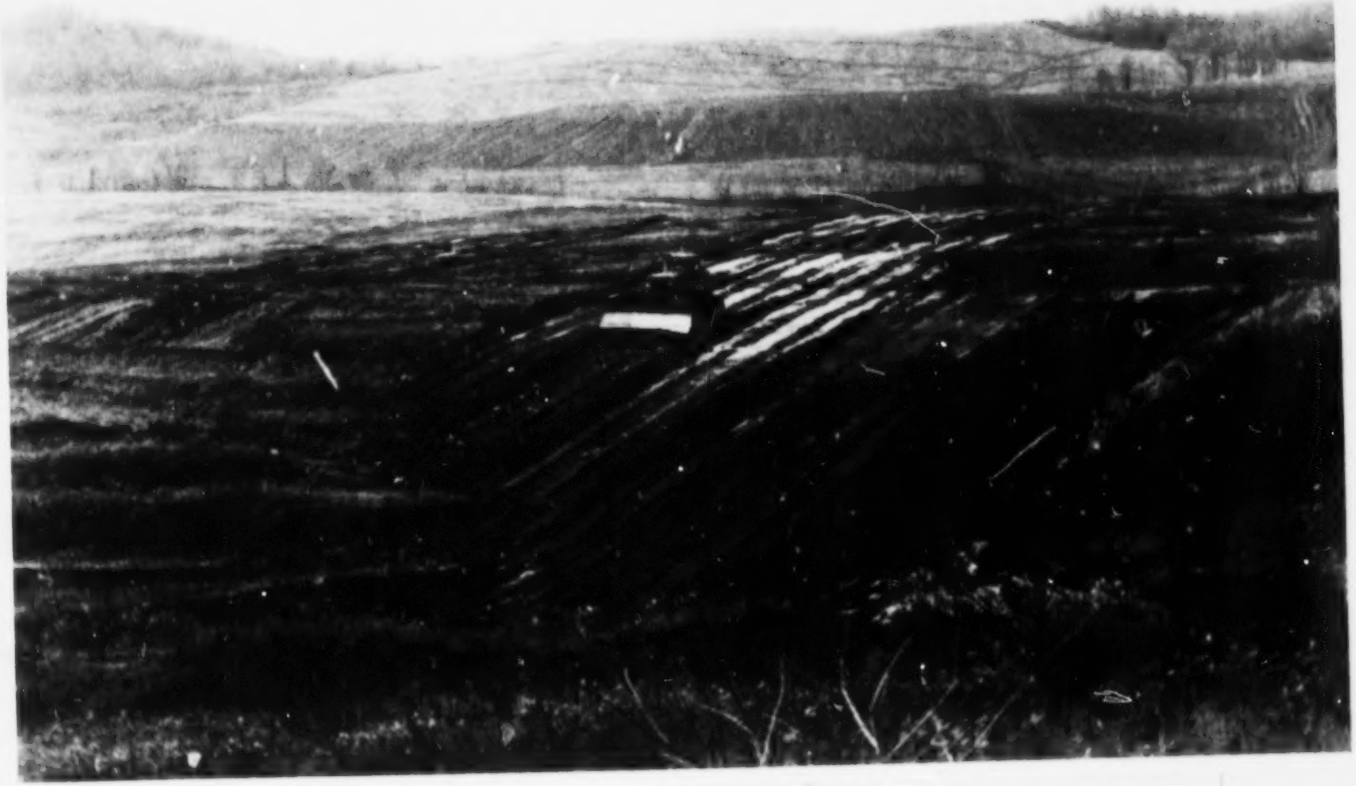

Figure 118. Bulldozer spreading topsoil on graded spoil.

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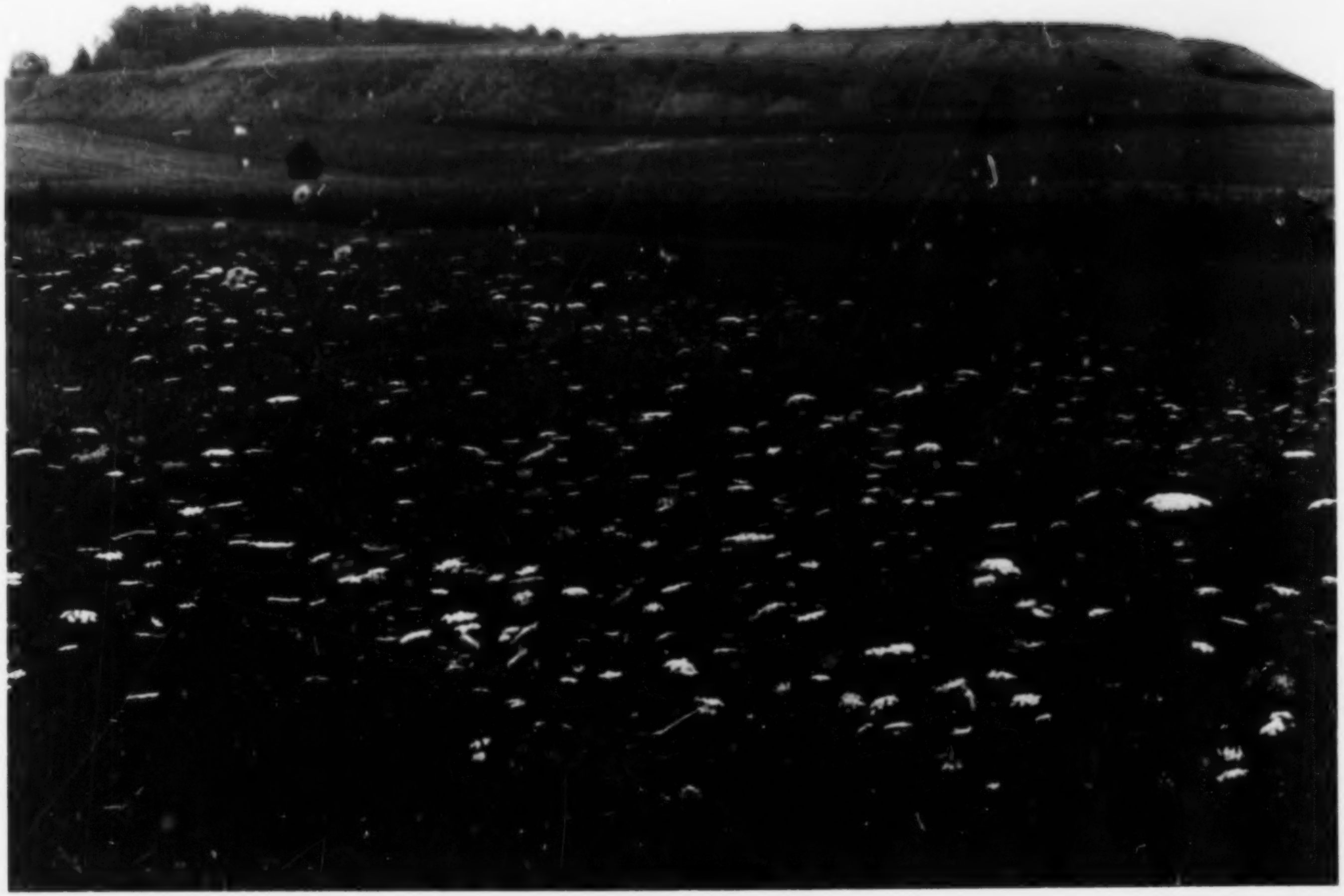

Figure 119. Coal mine spoil banks reclaimed to farmland.

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Figure 120. Area which was mined and reclaimed in 1975. The stripping equipment can be seen in the background (Photo taken in July 1976).

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SUMMARY

This chapter has briefly reviewed the premining planning for reclamation. Detailed discussion on water-management and land-use planning aspects were avoided in view of the extensive background reports on these two areas. A time sequence for reclamation activities was provided, along with examples of good reclamation practices.

SECTION 10

SURFACE MINING COSTS

INTRODUCTION

Economic evaluation of various mining plans is necessary to make a comparison of the alternatives. The key to preliminary analyses is the stripping ratio since, for a given situation, it usually dictates the initial decision to mine or not to mine by surface methods. Capital cost considerations can hardly be over-emphasized. However, equipment costs vary widely and are a function of the amount of steel and the fabrication in the design and construction of the equipment. Cost involved for other surface facilities (e.g., storage, office space, buildings, etc.) are also subject to great regional variances. Also, the financing and accounting procedures of companies differ, thereby making it difficult to arrive at meaningful strip mining costs.

Estimation of the mining costs must be, by necessity, based on the company's experience. The labor and material cost must be estimated for drilling, explosives, overburden removal, reclamation, pit cleaning, coal loading, haulage, road building, fuel, oil, grease, maintenance, supervision, depreciation, etc. Additionally, costs for transporting, erecting, dismantling, and moving the primary stripping and other equipment must be considered. Since the viability of a project must be determined over the mine life, these have to be projected into the future, taking into account the inflationary and productivity trends.

The selling price of coal is an uncertain factor. It is a complex function of the demand and availability of other energy resources and their prices. The correlation between the selling price and the mining and preparation cost, on one hand, and the attractiveness of investment in stripping, on the other, is strong.

STRIPPING RATIO

The stripping ratio is the proportion of overburden to coal removed. Though there are several units in which this proportion can be defined, the most common in the industry is in the *in situ* cubic yards of overburden removed per ton of clean coal (1 ton = 2000 lb). The equivalent unit in the metric system will be cubic meters of overburden removed per tonne (1000 kg) of clean coal. Simple arithmetic leads to the conversion factor between these two units:

$$\left(\frac{\text{m}^3 \text{ of overburden/}}{\text{tonne of coal}}\right) = \left(\frac{\text{yd}^3 \text{ of overburden/}}{\text{ton of coal}}\right) \times \left(\frac{1}{1.2075}\right)$$

The stripping ratio can also relate the selling price of coal with the costs of mining the coal and stripping the overburden. In technical literature, the calculations are sometimes based on average overburden depths, though in reality, the break-even stripping ratio is a point value beyond which the coal seam cannot be economically stripped. As the overburden depth increases, more money is spent on exposing the coal seam until a limit is reached when the value of the recovered material (clean coal) is just enough to pay for all the cost involved in mining, preparing, and selling the material.

The removal of overburden, or stripping, is generally regarded as the most significant component of surface coal mining costs. Variations in stripping ratio affect the scale of the equipment and the efficiency of its operation as far as overburden handling is concerned, while procedures for handling, preparing and marketing coal, and costs associated with these three steps are, in comparison, fixed from mine to mine.

Distinction is often made between several different ratio calculations, such as actual or overall stripping ratio, ultimate or break-even stripping ratio, daily or working stripping ratio, and the stripping ratio developed on the basis of underground mining costs vs. surface mining costs excluding stripping costs. The considerations of various ratios are also dependent on the scale of the operation, the capabilities of equipment and the general economic structure of the company.

Usually the first analysis is the calculation of a stripping ratio at which recovery by surface mining is cost equivalent to recovery by underground methods. Once a surface vs. underground ratio is established, the amount of surface mineable reserves is established. In addition, whatever resources can be recovered at a lower stripping ratio, it is being mined more economically than is possible by an underground approach.

For example, assume that the underground mining costs for a 1.0-m (3.3-ft) seam are \$8.50 per tonne (\$7.74 per ton). For the same seam, suppose the removal costs by surface methods are \$0.35 per tonne (\$0.32 per ton), and that the overburden can be removed by large equipment for \$0.20 per m (\$0.18 per yd^{3}). Costs for regrading and reclaiming the mined area are assumed to be \$700 per ha (\$0.07 per m^{2} or \$283 per acre). The underground vs. surface-stripping ratio (R_{su}) may be calculated as follows, realizing that there are 16145 tonnes/ha-m (1800 tons/acre-ft.):

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$$R_{su} = \frac{\text{Underground mining cost per tonne coal} - \text{Surface mining and reclamation cost per tonne coal}}{\text{Surface stripping cost per m}^3\text{ overburden}}$$

$$R_{su} = \frac{\$8.50 - ((\$0.35) + (\frac{\$700}{16145}))}{\$0.20}$$

$$R_{su} = \frac{\$8.50 - (\$0.35 + \$0.04)}{\$0.20}$$

$$R_{su} = 40.55\ \text{m}^3\ \text{overburden/tonne coal}$$

Coal should therefore be mined only in that portion of the deposit where the ratio does not exceed 40.55 m^3 overburden per tonne coal. In a deposit to be area mined, these calculations would show that the deposit could be exploited by surface mining as long as the overburden covering was less than 64.88m (197 ft). These conditions would be encountered in the Ohio eastern regions which incorporate the largest stripping equipment to such depths of overburden.

Total design of the mine requires that an ultimate break-even stripping ratio be calculated. This ratio is that which establishes the ultimate limits of mining, and is not the overall stripping ratio. The break-even ratio is that ratio at which the total cost of marketing one tonne of coal is equal to the price of that tonne of coal. The overall ratio must be less than the break-even ratio or no profit would be realized. The break-even ratio (R_{BE}) is calculated as:

$$R_{BE} = \frac{\text{Recoverable value per unit of coal} - \text{Production cost per unit of coal}}{\text{Stripping cost per unit waste}}$$

Working in common units of tonnes of coal and m^3 of overburden, a family of analyses and curves may be prepared based on various costs and prices which can be estimated.

Table 59 shows an analysis accounting for variations in percent reject and market price of coal.

In reality, R_{BE} (or any stripping ratio) is most commonly a linear polynomial function in several variables, including those as summarized in Table 60.

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TABLE 59. ANALYSIS OF STRIPPING RATIOS

PER CENT REJECT	5	10	15	20
Recovered Coal per mined tonne	0.950 tonne	0.900 tonne	0.85 tonne	0.80 tonne
Costs: Mining	$2.00 per tonne			
Washing	2.00 per tonne			
Reclamation	0.05 per tonne			
Marketing	6.50 per tonne			
Total Cost, excluding stripping	$10.55 per tonne			
Given stripping cost =	$0.20 per m^3			
For market value of				
$15 per tonne –				
Value =	$14.25	$13.50	$12.75	$12.00
Net =	3.70	2.95	2.20	1.45
Ratio =	18.5:1	11.75:1	11.0:1	7.25:1
$30 per tonne –				
Value =	$28.50	$27.00	$25.50	$24.00
Net =	17.95	16.45	14.95	13.45
Ratio =	89.75:1	82.25:1	74.75:1	67.25:1
$45 per tonne –				
Value =	$42.75	$40.50	$38.25	$36.00
Net =	32.20	29.95	27.70	25.45
Ratio =	161:1	149.75:1	138.5:1	127.25
$60 per tonne –				
Value =	$57.00	$54.00	$51.00	$48.00
Net =	46.45	43.45	40.45	37.45
Ratio =	232.25:1	217.25:1	202.25:1	187.25

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TABLE 60. FACTORS IN STRIPPING RATIO CALCULATIONS

FACTOR	SYMBOL	TYPICAL RANGE(1)	
		LOW	HIGH
Percent reject, as decimal	P_{RE}	0	60
Mining cost per tonne mined	C_{MI}	\$ 0.20	\$ 2.00(2)
Cleaning cost per tonne	C_{CL}	\$ 0.00	\$ 5.00
Site preparation and revegetation cost per tonne mined	C_{RE}	\$ 0.003	\$ 1.00
Stripping cost per cubic meter overburden	C_{ST}	\$ 0.04	\$ 1.00
Market price of coal, per clean tonne	P_{CC}	\$10.00	\$100.00(3)
Transportation, sales and overhead per clean tonne	O_{HD}	\$ 0.00(4)	\$ 15.00

Notes

1. Although some values may appear high at the present time, allowances are made for trends in the future.
2. Some mines do encounter long (>10 km) truck haulage, which is considered as secondary transportation in this analysis and included in O_{HD}.
3. Market price for metallurgical coal could conceivably inflate to this level in the future.
4. Mine-mouth plants have essentially no O_{HD} costs.

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The break-even stripping ratio function may be expressed as:

$$R_{BE} = \frac{(P_{CC} - O_{HD}) \times (1.0 - P_{RE}) - (C_{MI} + C_{CL} + C_{RE})}{C_{ST}}$$

or, $$R_{BE} = \frac{P_{CC}}{C_{ST}} - \frac{(P_{RE})(P_{CC})}{C_{ST}} + \frac{(O_{HD})(P_{RE})}{C_{ST}} - \frac{C_{MI}}{C_{ST}} - \frac{C_{CL}}{C_{ST}} - \frac{C_{RE}}{C_{ST}} - \frac{O_{HD}}{C_{ST}}$$

In this second form, it is apparent that the calculation of a break-even stripping ratio may be performed by determining each term separately and accumulating the result.

A sample calculation is displayed as follows:

Given: P_{RE} = 0.10 (i.e., 10% reject)
C_{MI} = \$ 1.00 per tonne coal
C_{CL} = \$ 3.00 per tonne coal
C_{RE} = \$ 0.35 per tonne coal
O_{HD} = \$ 8.00 per clean tonne coal
P_{CC} = \$22.50 per clean tonne coal
C_{ST} = \$ 0.25 per cu m overburden coal

then,

$$\begin{array}{lll}
P_{CC} \times (1. - P_{RE}) \div C_{ST} & = \$22.50 \times 0.9 \div 0.25 = & +\ 81.0\ m^3/t \\
-O_{HD} \times (1. - P_{RE}) \div C_{ST} & = \$\ 8.00 \times 0.9 \div 0.25 = & 28.8\ m^3/t \\
-C_{MI} \div C_{ST} & = \$\ 1.00 \div 0.25 \quad = & 4.0\ m^3/t \\
-C_{CL} \div C_{ST} & = \$\ 3.00 \div 0.25 \quad = & 12.0\ m^3/t \\
-C_{RE} \div C_{ST} & = \$\ 0.35 \div 0.25 \quad = & 1.4\ m^3/t \\
\hline
R_{BE} & = \text{Total} & +\ 34.8\ m^3/t
\end{array}$$

It is possible to analyze variations in stripping ratio and sensitivity with regard to variations in any single factor or combination of factors. Rather than attempt such an involved analysis, the following figure presents

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the variation in stripping ratio with respect to the total net value per tonne of coal, excluding of stripping cost, and stripping cost per cubic meter of overburden. Figure 121 shows the stripping ratio in cubic meters of overburden per tonne of coal. It is possible to obtain the stripping ratio in meters of overburden per meter of coal by multiplying this ratio by 1.34. Of course, not all of these situations are probable in today's conditions. However, even the unlikely costs of $100 for coal and $0.05 for stripping could be possible if high-grade metallurgical coal continues to rise in price, while earth-moving technology continues to develop along its current trends.

STRIPPING RATIO IN MULTIPLE-SEAM MINING

The analysis of stripping ratio may be complicated by considering the case where several seams are mined from one cut. Multiple-seam planning requires that both total stripping ratio and the incremental stripping ratios be analyzed. A necessary condition is that the cost of stripping all overburden, innerburden, and partings be less than or equal to the total net value of coal extracted from the seams. However, the total analysis is not, of its own, sufficient to justify multiple-seam mining. Even if it is feasible to remove two seams in terms of the overall stripping ratio, the lower seam will not be mined unless the incremental stripping ratio for the lower seam is also acceptable.

Consider, for example, a situation where the deposit consists of 10 m of overburden, 1.0 m of coal, 20 m interburden, and 0.5 m coal. The overall stripping ratio is calculated as:

$$\frac{10 + 20 \text{ m overburden}}{1.0 + 0.5 \text{ m coal}} = \frac{30}{1.5} = 20{:}1, \text{ m/m}$$

If the break-even ratio for the company were 25:1, m/m, then the total ratio would be acceptable, and seemingly both seams should be mined. However, the incremental stripping ratios are:

$$\text{Upper seam, } \frac{10 \text{ m}}{1\text{m}} = 10{:}1 \text{ m overburden to m coal}$$

$$\text{Lower seam, } \frac{20 \text{ m}}{0.5 \text{ m}} = 40{:}1 \text{ m interburden to m coal}$$

While the upper seam is quite favorable for mining, the incremental analysis of the lower seam fails to justify mining both seams.

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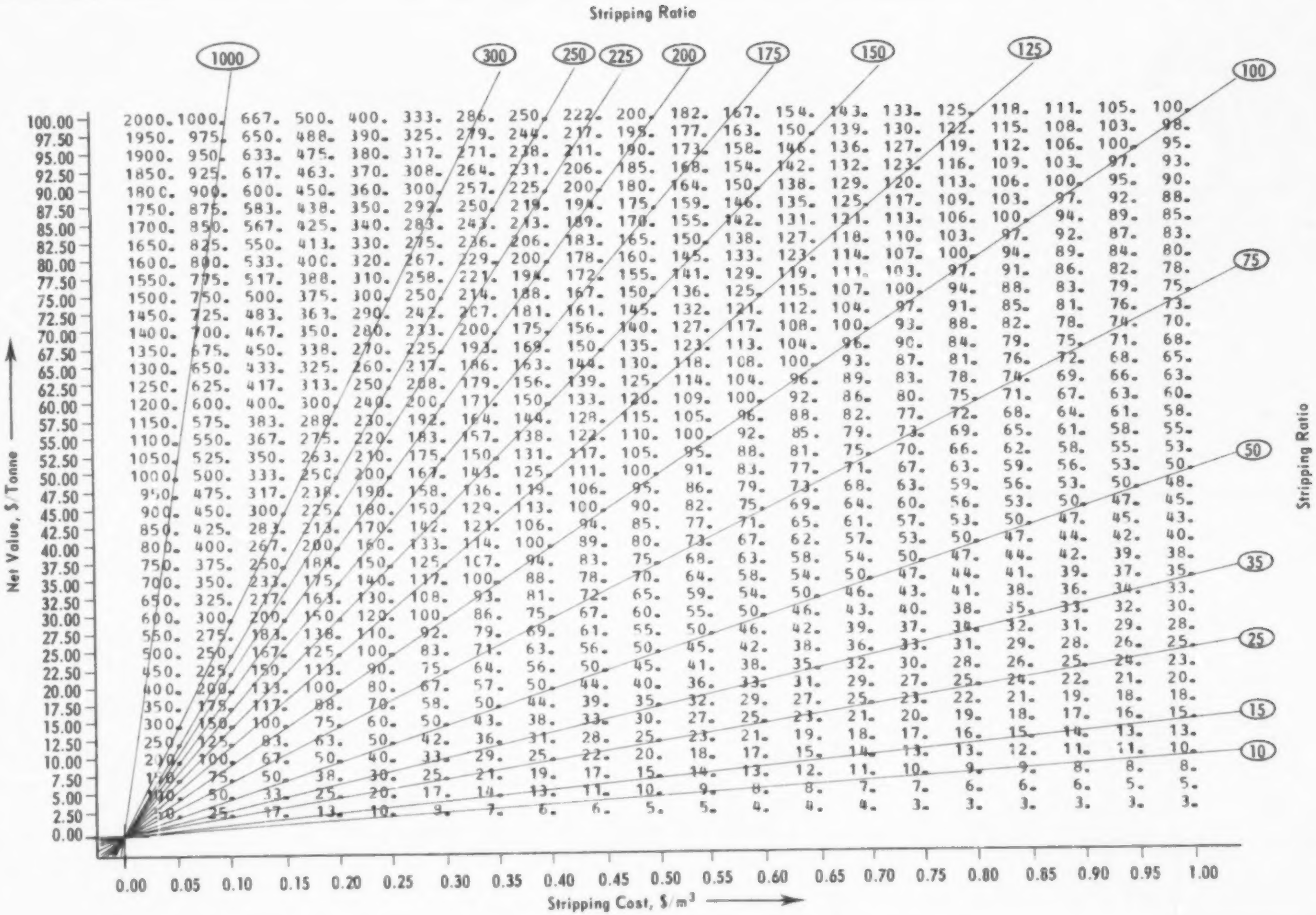

Net Value, $/Tonne	0.05	0.10	0.15	0.20	0.25	0.30	0.35	0.40	0.45	0.50	0.55	0.60	0.65	0.70	0.75	0.80	0.85	0.90	0.95	1.00
100.00	2000.	1000.	667.	500.	400.	333.	286.	250.	222.	200.	182.	167.	154.	143.	133.	125.	118.	111.	105.	100.
97.50	1950.	975.	650.	488.	390.	325.	279.	244.	217.	195.	177.	163.	150.	139.	130.	122.	115.	108.	103.	98.
95.00	1900.	950.	633.	475.	380.	317.	271.	238.	211.	190.	173.	158.	146.	136.	127.	119.	112.	106.	100.	95.
92.50	1850.	925.	617.	463.	370.	308.	264.	231.	206.	185.	168.	154.	142.	132.	123.	116.	109.	103.	97.	93.
90.00	1800.	900.	600.	450.	360.	300.	257.	225.	200.	180.	164.	150.	138.	129.	120.	113.	106.	100.	95.	90.
87.50	1750.	875.	583.	438.	350.	292.	250.	219.	194.	175.	159.	146.	135.	125.	117.	109.	103.	97.	92.	88.
85.00	1700.	850.	567.	425.	340.	283.	243.	213.	189.	170.	155.	142.	131.	121.	113.	106.	100.	94.	89.	85.
82.50	1650.	825.	550.	413.	330.	275.	236.	206.	183.	165.	150.	138.	127.	118.	110.	103.	97.	92.	87.	83.
80.00	1600.	800.	533.	400.	320.	267.	229.	200.	178.	160.	145.	133.	123.	114.	107.	100.	94.	89.	84.	80.
77.50	1550.	775.	517.	388.	310.	258.	221.	194.	172.	155.	141.	129.	119.	111.	103.	97.	91.	86.	82.	78.
75.00	1500.	750.	500.	375.	300.	250.	214.	188.	167.	150.	136.	125.	115.	107.	100.	94.	88.	83.	79.	75.
72.50	1450.	725.	483.	363.	290.	242.	207.	181.	161.	145.	132.	121.	112.	104.	97.	91.	85.	81.	76.	73.
70.00	1400.	700.	467.	350.	280.	233.	200.	175.	156.	140.	127.	117.	108.	100.	93.	88.	82.	78.	74.	70.
67.50	1350.	675.	450.	338.	270.	225.	193.	169.	150.	135.	123.	113.	104.	96.	90.	84.	79.	75.	71.	68.
65.00	1300.	650.	433.	325.	260.	217.	186.	163.	144.	130.	118.	108.	100.	93.	87.	81.	76.	72.	68.	65.
62.50	1250.	625.	417.	313.	250.	208.	179.	156.	139.	125.	114.	104.	96.	89.	83.	78.	74.	69.	66.	63.
60.00	1200.	600.	400.	300.	240.	200.	171.	150.	133.	120.	109.	100.	92.	86.	80.	75.	71.	67.	63.	60.
57.50	1150.	575.	383.	288.	230.	192.	164.	144.	128.	115.	105.	96.	88.	82.	77.	72.	68.	64.	61.	58.
55.00	1100.	550.	367.	275.	220.	183.	157.	138.	122.	110.	100.	92.	85.	79.	73.	69.	65.	61.	58.	55.
52.50	1050.	525.	350.	263.	210.	175.	150.	131.	117.	105.	95.	88.	81.	75.	70.	66.	62.	58.	55.	53.
50.00	1000.	500.	333.	250.	200.	167.	143.	125.	111.	100.	91.	83.	77.	71.	67.	63.	59.	56.	53.	50.
47.50	950.	475.	317.	238.	190.	158.	136.	119.	106.	95.	86.	79.	73.	68.	63.	59.	56.	53.	50.	48.
45.00	900.	450.	300.	225.	180.	150.	129.	113.	100.	90.	82.	75.	69.	64.	60.	56.	53.	50.	47.	45.
42.50	850.	425.	283.	213.	170.	142.	121.	106.	94.	85.	77.	71.	65.	61.	57.	53.	50.	47.	45.	43.
40.00	800.	400.	267.	200.	160.	133.	114.	100.	89.	80.	73.	67.	62.	57.	53.	50.	47.	44.	42.	40.
37.50	750.	375.	250.	188.	150.	125.	107.	94.	83.	75.	68.	63.	58.	54.	50.	47.	44.	42.	39.	38.
35.00	700.	350.	233.	175.	140.	117.	100.	88.	78.	70.	64.	58.	54.	50.	47.	44.	41.	39.	37.	35.
32.50	650.	325.	217.	163.	130.	108.	93.	81.	72.	65.	59.	54.	50.	46.	43.	41.	38.	36.	34.	33.
30.00	600.	300.	200.	150.	120.	100.	86.	75.	67.	60.	55.	50.	46.	43.	40.	38.	35.	33.	32.	30.
27.50	550.	275.	183.	138.	110.	92.	79.	69.	61.	55.	50.	46.	42.	39.	37.	34.	32.	31.	29.	28.
25.00	500.	250.	167.	125.	100.	83.	71.	63.	56.	50.	45.	42.	38.	36.	33.	31.	29.	28.	26.	25.
22.50	450.	225.	150.	113.	90.	75.	64.	56.	50.	45.	41.	38.	35.	32.	30.	28.	26.	25.	24.	23.
20.00	400.	200.	133.	100.	80.	67.	57.	50.	44.	40.	36.	33.	31.	29.	27.	25.	24.	22.	21.	20.
17.50	350.	175.	117.	88.	70.	58.	50.	44.	39.	35.	32.	29.	27.	25.	23.	22.	21.	19.	18.	18.
15.00	300.	150.	100.	75.	60.	50.	43.	38.	33.	30.	27.	25.	23.	21.	20.	19.	18.	17.	16.	15.
12.50	250.	125.	83.	63.	50.	42.	36.	31.	28.	25.	23.	21.	19.	18.	17.	16.	15.	14.	13.	13.
10.00	200.	100.	67.	50.	40.	33.	29.	25.	22.	20.	18.	17.	15.	14.	13.	13.	12.	11.	11.	10.
7.50	150.	75.	50.	38.	30.	25.	21.	19.	17.	15.	14.	13.	12.	11.	10.	9.	9.	8.	8.	8.
5.00	100.	50.	33.	25.	20.	17.	14.	13.	11.	10.	9.	8.	8.	7.	7.	6.	6.	6.	5.	5.
2.50	50.	25.	17.	13.	10.	9.	7.	6.	6.	5.	5.	4.	4.	4.	3.	3.	3.	3.	3.	3.
0.00																				

Figure 121. Stripping ratio in m^3/tonne for various coal values and stripping costs.

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A complication might exist when considering more than two seams; a situation where mining the upper two seams is not feasible while mining three or more seams is feasible. Consider a situation as above, but in which 10m of interburden lie over a third, lowest seam, 2.5 m thick. The incremental analysis would become:

Overall, $\frac{10 + 20 + 10}{1 + 0.5 + 2.5} = 10:1$ m/m

Upper seam, 10:1, as above

Middle seam, 40:1, as above

Lowest seam, without considering middle seam

$$\frac{20 + 0.5 + 10}{2.5} = \frac{30.5}{2.5} = 12.2:1, \text{ m/m acceptable}$$

Lowest seam, including middle seam as byproduct

$$\frac{20 + 10}{3.0} = 10:1$$

It would then be advisable to mine to the lowest seams. Of course, since the lowest seam is to be mined, and since the stripping ratio considering the second seam as "byproduct" is better than mining the lowest alone, the second seam will also be recovered. Unique cases may exist when the particular economics of coal handling and cleaning may indicate that the middle seam should be treated as interburden. With stringent requirements for the burial of refuse and conservation of natural resources, the practice may not be justified.

STRIPPING RATIO AND COSTS

The determination of stripping ratio has been shown to be quite dependent upon and sensitive to a variety of costs. Such a dependence on costs means that the break-even stripping ratio is related to the scale of the operation. The scale and type of operation, economic nature of the company, and reclamation approach will affect the development of the break-even stripping ratio and the daily or working stripping ratio of the operation.

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The working stripping ratio is generally much less than the break-even ratio. It varies with the depth of cover at the time and provides an overall ratio less than or equal to the break-even stripping ratio. In general, lower stripping ratios imply lower production costs. An objective in planning is often to mine with the contour. Mining with the contour means starting where the depth of overburden is least or at the outcrop, if the seam is outcropping. The working stripping ratio then increases with the progress of mining. Such an approach allows higher unit profits (i.e. low production costs) in the first years of life, when the time value of money is highest, and defers high unit production costs to a later point in time. Additionally, if improvements in technology or a higher price for the product is attained at a future date, the break-even stripping ratio can be increased. However, where the topography is severe (steep hill), the break-even point is reached in a very small width. Therefore, the entire width is mined to the break-even point and then reclaimed. However, in these operations, scale and bonding are minimized.

In area mining and mountaintop mining there is usually some coal recovered under working stripping ratios in excess of the break-even ratio. To be justified, the incremental gain in total profits attributable to mining the coal so burdened must still balance to an overall ratio less than the break-even ratio. This practice is required in the area directly under a mountaintop or under a small knoll in an area mine where leaving the mountaintop or knoll would require that different equipment of smaller scale be used in the entire operation. The extraction of a small tonnage of coal which does not, strictly speaking, pay for itself may be justified on the basis of economy of scale that could not be otherwise realized.

COST ANALYSIS

Determination of mining acceptability is not completely restricted to stripping ratio. There can also be established a minimal mineable thickness, a minimal value of coal, and technological limits to exploitation.

Where sufficient tonnage and market are indicated, a more detailed analysis is performed. In this analysis, specific alternative mining methods or groups of subsystems will be specified as indicated by the geometry of the deposit and the stripping ratio. The design problem can be one of selection of primary earthmoving methods and equipment from several alternatives. For each alternative method proposed, capital and operating expenses are detailed and more exact stripping limits and recoverable tonnages can be determined. There are at least three stages of analysis which are carried out:

1. Mine vs. no-mine analysis, based on stripping ratio, gross cost, and gross return estimates.
2. Selection and analysis of alternative methods, based on gross geological aspects.
3. Final design analysis, based on detailed comparison of specific equipment, production requirements, and legal and social constraints.

Costs incurred by mining companies are broken down in two ways: 1.) capital costs are large outlays of money for land, equipment, and buildings. For tax purposes, these costs are not regarded as directly deductible expenses, but must be either amortized, depreciated or depleted. 2.) operating costs (or expenses) are for such items as labor and supplies, and are deductible, in total, in the year of occurrence. Costs are also treated as direct or indirect, with fixed or variable costs, depending on their relationship to coal production.

Some capital expenditures are usually expected by the government to be treated as amortizable items. These include organization costs of the corporation under certain conditions, acquisitions, or nonmineral leases, research and development expenses not treated as expenses, amounts other than purchase connected with acquisition of trademarks, copyrights and the like, and the cost of certified pollution control facilities placed in service in the years 1969 through 1974, inclusive. Amortization is calculated by a straight line method. For example, for an amortizable item of five-year life, one fifth of the amount is deducted from the gross income in each year of the project life.

Depreciation is an allowable deduction prior to taxes for the wear and tear of equipment during their normal use. It is in effect a measure of the decrease in value of capital equipment items, and book value decreases as equipment depreciates with time. In any instance, depreciation is regarded as the yearly decrease in real value of property which is used or employed in the course of business. Depreciation may be calculated strictly on the basis of time or may be calculated strictly on the basis of units of production. By either method, the first step is to determine the depreciable amount for the item. This is generally the purchase price less the anticipated salvage value at the end of a useful life. While mining companies must establish meaningful and useful lives for buildings, machinery, and other depreciable items, sometimes guidelines and acceptable lives are established by the Internal Revenue Service (IRS). The purchase price, salvage value, and equipment life must be consistent with the IRS guidelines and acceptable to the IRS.

A company exploiting a wasting asset like coal is allowed to deduct a certain sum from the income it receives from mining coal to reflect the loss of the capital asset, which is the coal in the ground. This sum is the depletion allowance. Two alternative methods are available for calculating this depletion allowance, cost depletion and percentage depletion. The company can select whichever method gives the greater deduction and is also allowed to switch from one to another, if desired, from year to year. Methods of depreciation and depletion are discussed in several engineering economy texts.

The operating costs can be broken down into direct and indirect, and fixed and variable costs. Certain items of the mining work can be directly related to the production of a given number of units of coal; the amount of work the men in the pit perform with the utilization of a shovel or dragline. Other costs, such as taxes, sewer, mine plant maintenance, dry-change facilities, etc., do not relate directly to the production of specific units of coal but accrue indirectly to the operation as a whole. The former may be regarded as direct costs while the latter are regarded as indirect costs. In a similar analysis, certain costs are fixed without regard to the quantity of coal being mined, such as the maintenance of facilities, engineering staff pay, supervisory pay, etc. Other costs, such as direct labor pay, operation expenses for drills, shovels, trucks, blasting, etc., are incurred directly by and can be accounted directly to the production of given quantities of coal. In most cases, these fixed costs are prorated for accounting purposes to a unit of production basis.

Where appropriate accounting and industrial engineering records have been kept, the estimation of costs for new methods and new approaches is greatly facilitated, as the increments of cost involved in the new approach may well be historically documented at present in previous, traditional approaches.

COST ESTIMATION

It is not the purpose here to go into details of cost analysis models. From the discussion thus far, it is obvious that there is no one model or method that will give correct answers. Even if the models or analysis procedures are analytically sound, the input data in most cases is proprietary to the company. The U.S. Bureau of Mines has published several models that address the cost of surface mining of coal.

Boyce (1975) reports on a computerized model to analyze the technical and financial aspects of large-scale surface mining. The model is intended to be used for first-order planning of large scale surface coal mines and for cost evaluation under various conditions. It was developed for the Energy Research and Development Administration (ERDA) by Flour Utah, Incorporated.

Otte and Boehlje (1975) have developed a model for calculating the cost of mining and reclamation with various materials handling techniques and procedures in different pit configuration. This computer-oriented model calculates production and costs for each individual task performed to uncover and remove coal and to reclaim the mined area. Total costs per ton of coal removed and the capital outlay required to obtain a specified level of annual production are also determined by aggregating the cost of each task.

Manual et al (1975) have developed a cost-model for anthracite open-pit mining which determines a rate of return for a specified realization (i.e., selling price) or vice-versa through a discounted cash-flow analysis. The model divides costs into four groups; ownership, labor, supplies, and miscellaneous costs. The last category includes the charges for welfare, royalties, etc.

NUS Corporation has developed a cost model for the Electric Power Research Institute (EPRI, 1977). The model incorporates the essential features of surface coal mine equipment selection and cost engineering

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principles to analyze the costs of producing coal from a new surface mine. It is intended to provide an engineering approach to determination of production costs to be used in developing coal price-availability relationships.

A computer model has been developed by Ramani, Murray and Manula (1977) to generate the selling price of coal based on a combination of geologic, engineering and economic variables, as these are related to area, contour and open-pit mining. Much of the background information used in model development was based on the report by the NUS Corporation to Electric Power Research Institute (EPRI, 1977).

Surface Mining Cost Models

Katell and Hemingway (1972, 1976) have estimated typical capital investments, operation costs, and selling prices for coal strip mines that will fuel a 250-million-standard-cubic foot-per-day, high-BTU gas plant in Eastern, Interior, and Northern Great Plains states of the United States. These studies are regularly updated; the latest study can be found in the Bureau of Mines Information Circular 8703 (Katell, Hemingway and Berkshire, 1976).

Toth and Stinnett (1976) present a somewhat simple model of the NUS/EPRI Model (EPRI, 1977) for a preliminary estimate of surface mining costs. The model is presented here in its entirety because the principles on which the model is based can easily be extended to any surface mining operation with modifications. The accuracy of the model results may be in the range of ± 30%, typical of preliminary estimates. Since some values used here are outside the range of the graphs, instead of using the graphs, calculations are performed to arrive at the parameter values. In general, the approach presented here parallels that which an engineer would follow in performing a preliminary new mine cost estimate. The "bottom line" of the analysis is the minimum, acceptable selling price for the coal (from the seller's viewpoint), but certain other important figures are generated as well. The model centers around a set of nine graphs which allow for gross variances that are usually encountered among coal deposits, and some input, therefore, is required from the user.

The first graph (Figure 122) illustrates the relationship between the designed mine capacity and the volume of overburden which must be removed. The variable relating the two is the stripping ratio. In actual practice, a mine's stripping ratio will vary from year to year and likely will approach some upper limit late in life, beyond which the mine can no longer be operated profitably. The strip ratio suggested for this analysis is the average effective strip ratio (including rehandle), although a rough sensitivity analysis can be performed by simply varying the ratio selected and noting the effect on coal selling price. A 1,696,720 tonne/year (1,670,000 ton/year) mine located in Illinois and characterized by a strip ratio of twelve cubic yards of overburden per ton of coal is selected for illustrative purposes. The resulting overburden figure, 15,290,000 cu m (20,000,000 cubic yards), is used as input to the next graph (Figure 122).

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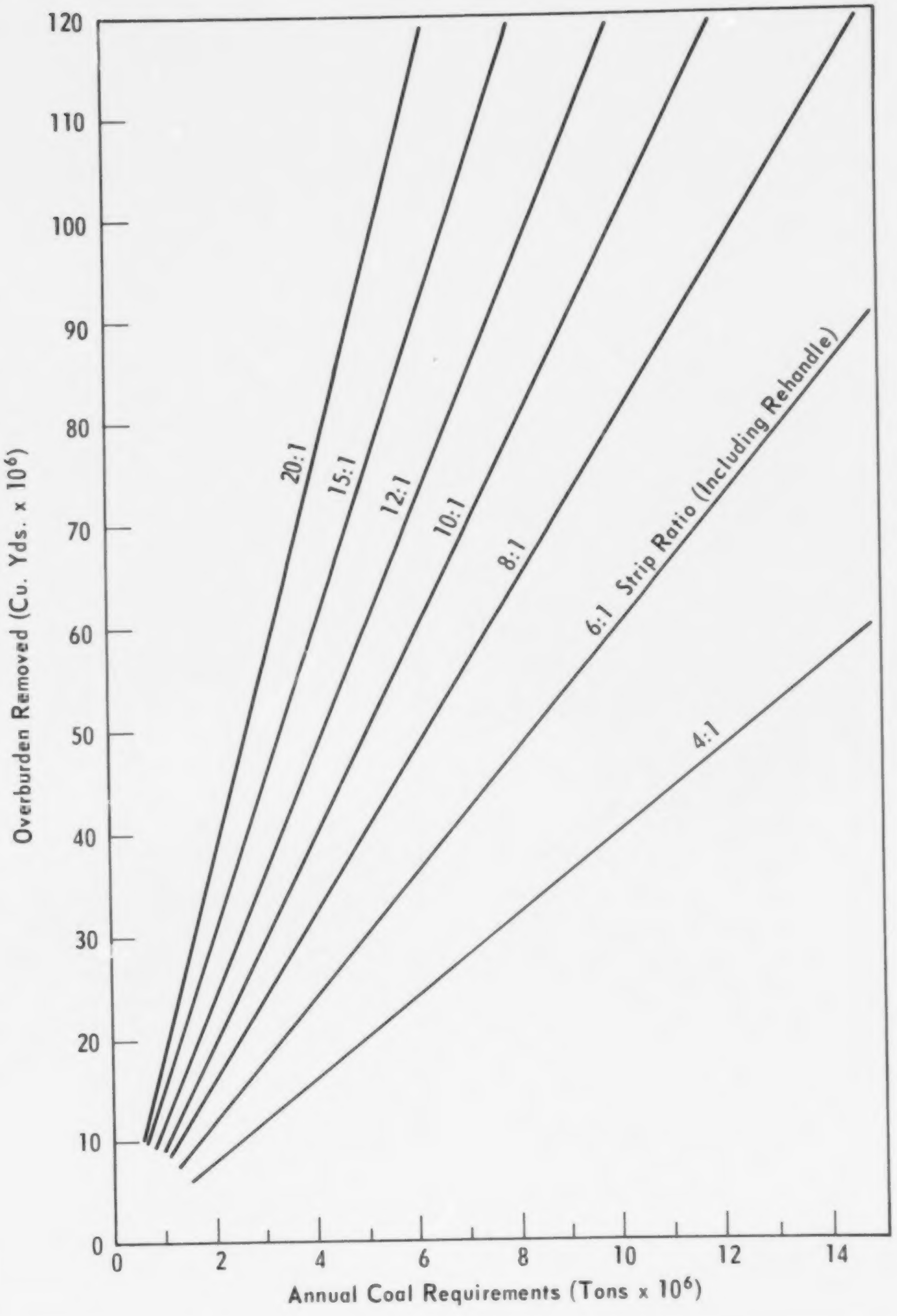

Figure 122. Production versus overburden removal.

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In Figure 123, the annual overburden to be removed is translated into total machine capacity needed to affect removal. The basic overburden removing equipment in area stripping operations is assumed to be either a shovel or dragline. The variable in Figure 123 is machine "efficiency" which is dependent upon a number of operating parameters including cycle time, operating hours, operator efficiency, maintenance reliability and so forth. Additionally, if the machine is used for road building or encounters excessive rehandling or chopping down as a result of multiple-seam mining, the efficiency will be lowered considerably.

In the example, overburden digging conditions are presumed to be good--blasting will not likely be required--and a single seam will be extracted. Thus selecting a moderately high efficiency of 366,252.45 cu m/cu m (280,000 yards/yard) of bucket capacity, 53.51 cu m (70 cubic yards) of machine capacity will be needed.

Figure 124 illustrates the relatively large component of capital costs attributable to the overburden stripping equipment, and thus suggests the rationale for focusing on the strip ratio as a primary factor in ultimately affecting the selling price of coal. There is reason to believe that machine costs may not exhibit the straight line function with size as is illustrated; however, since an operator may have the option of selecting multiple pieces of smaller capacity to meet production needs, the straight line assumption should remain with the accuracy of this analysis.

The three other lines on the graph show typical capital outlay for physical investment (the proven reserves, initial equipment, and plant needs), the start-up mine investment (including interest during construction, preproduction development, and working capital), and the total investment throughout the mine operating life (including equipment replacements).

Referring to the example, the start-up and total mine investment may approach $48,000,000 and $73,000,000, respectively.

Using the total capital investment from the preceding graph, the straight line depreciation and amortization charges per ton of coal produced can be determined in Figure 125. All mines are presumed to have a 30-year operating life, although a similar chart could be constructed if this simplification is invalid. The number derived from Figure 125 represents the noncash portion of operating costs and will be summarized with others to arrive at the minimum acceptable selling price. Since the data for the mine is outside the limits of the graph in the example for this 1,696,720 tonne (1,670,000 ton/year) operation, per ton depreciation and amortization charges can be calculated as:

$$\frac{\$73,000,000}{30 \times 1,696,720} = \$1.43/\text{tonne} \ (\frac{\$73,000,000}{30 \times 1,670,000} = \$1.46/\text{ton})$$

This calculation also illustrates the simplicity of the use of the model.

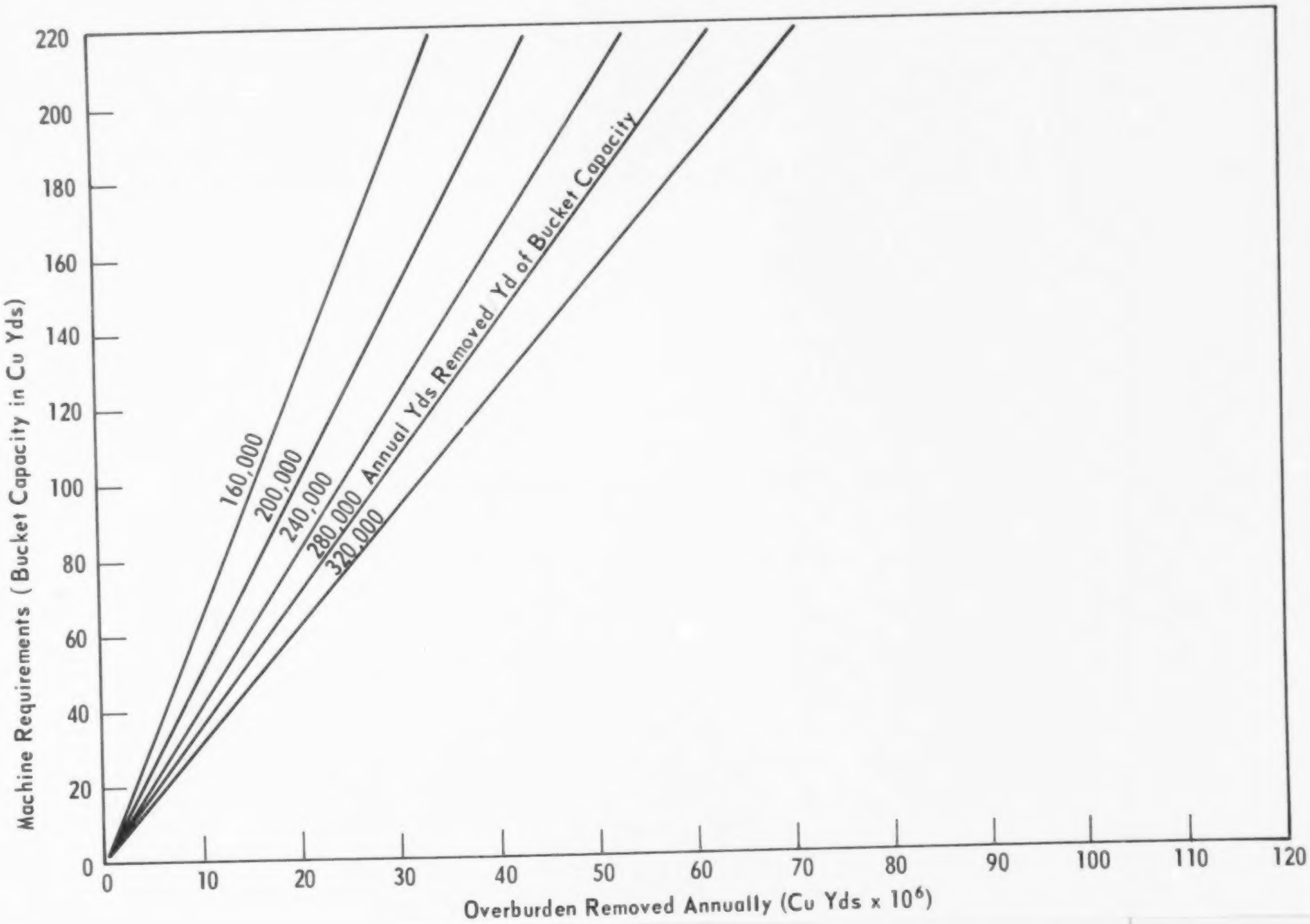

Figure 123. Overburden removal versus machine capacity.

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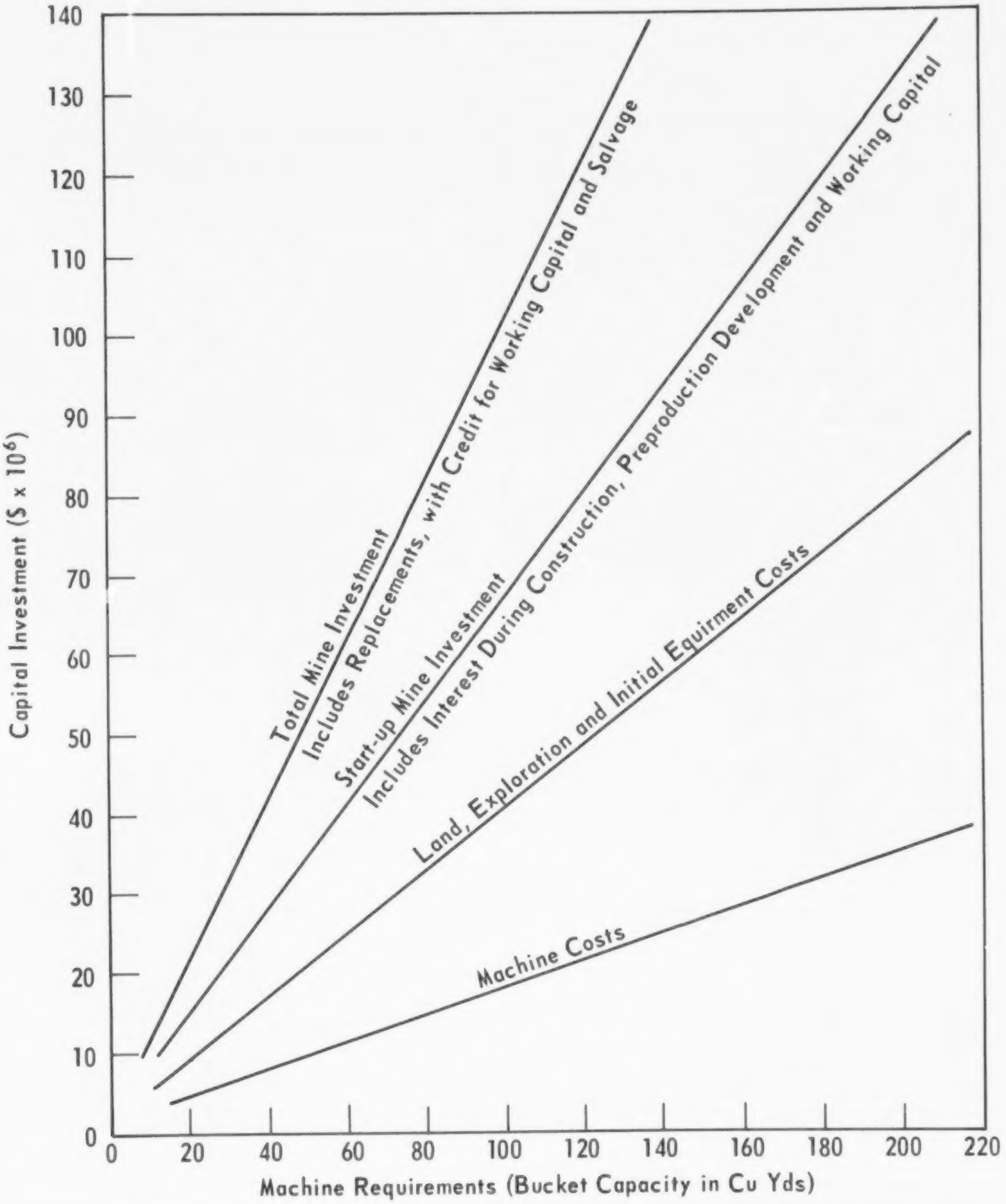

Figure 124. Capital requirements for a surface coal mine.

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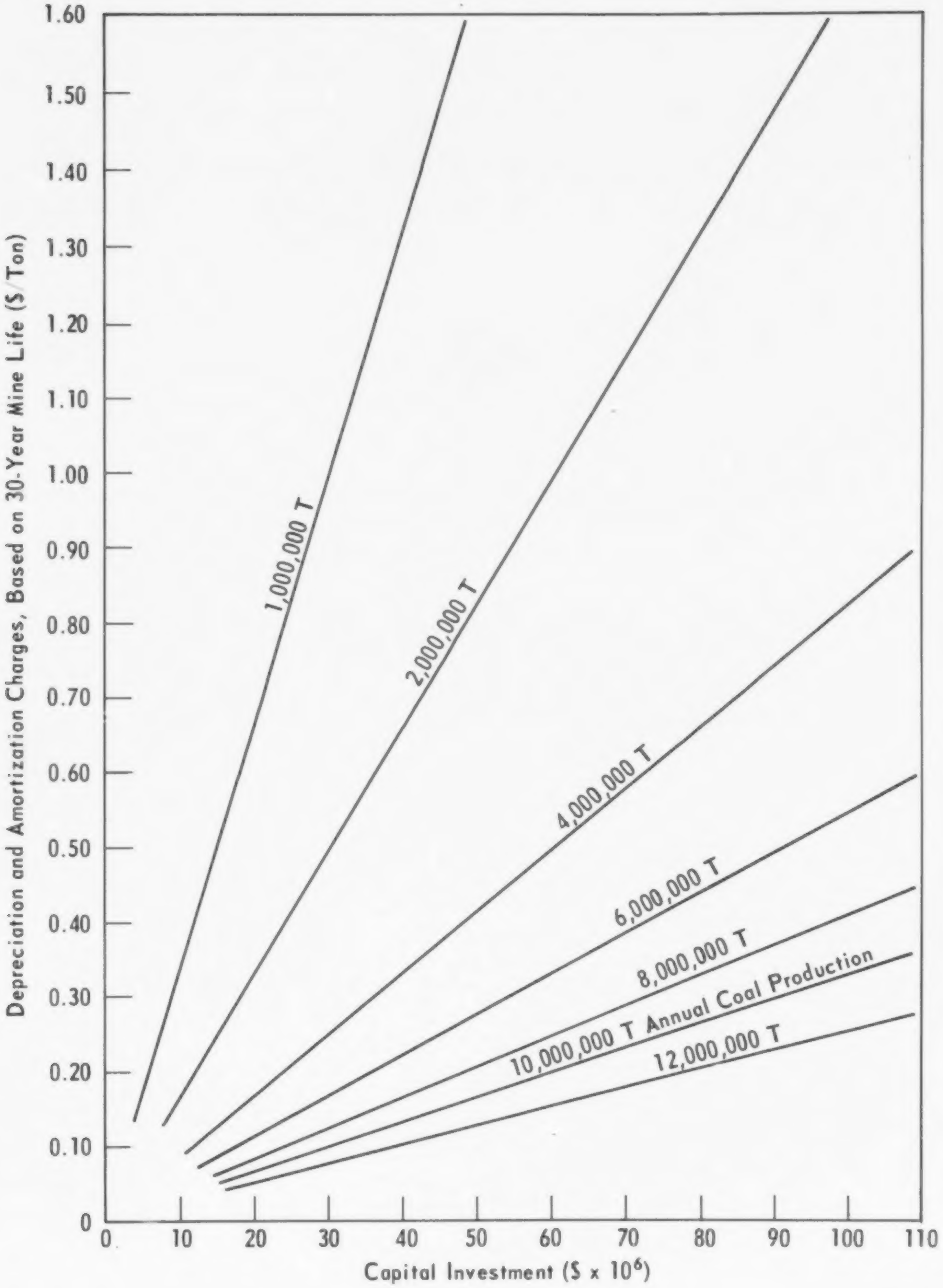

Figure 125. Graph for the computation of depreciation charges.

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Figure 126 is a display of the range and average productivity of surface mining operations in the United States, measured in tons/man-day. The range in production is by county, rather than individual mine, so presumably a range chart based on the latter category would show even greater variation.

The relatively low productivity of the entire United States of 36.57 tonne/man-day (36 tons/man-day) stems from the large proportion of coal coming from the Appalachian and Interior Regions. It is reasonable to assume in the example, however, that at a large new well-engineered mine in Illinois the labor productivity will be 30.48 tonne/man-day (30 tons/man-day).

Figure 127 relates the labor cost per ton of coal produced to the average labor rate and the productivity of the mine personnel. Mine personnel in this analysis includes supervisory staff in addition to all hourly workers.

Assuming a labor rate of \$60/day and a productivity of 30.48 tonne/man-day (30 tons/man-day), for the example, labor costs are determined to be \$2.26/tonne (\$2.3/ton). It is quite possible to vary both productivity and labor rate, and arrive at different labor costs.

Assuming a productivity of 30.48 tonne/man-day (30 tons/man-day) and approximately 250 working days/year, the minimum number of men not including provisions for absenteeism, vacations and turnover is calculated below.

$$\text{No. of men required} = \frac{\text{Annual Production}}{\text{(Productivity)(No. of working days/year)}}$$

$$\frac{1{,}696{,}720}{30.48 \times 250} = 223 \text{ employees or } \left(\frac{1{,}670{,}000}{30 \times 250} = 223 \text{ employees}\right)$$

Figure 128 provides the relationship between labor costs per ton of coal produced and direct operating costs for the operation. As defined here, direct operating costs include not only labor charges but also supplies, power, fringe benefits, welfare payments, insurances, royalties, property and payroll taxes, and so forth. Because of the catch-all nature of this grouping, and because of the initial premise that new, highly productive mines are to be considered for analysis, the proportion of labor costs to direct operating charges is predicated to be somewhat lower than historical records usually indicate.

The data for the mine is outside the limits of the graph. Therefore, with labor charges estimated at 30% of total direct charges, the direct operating costs are calculated to be \$7.54 per tonne (\$7.67 per ton) of coal (i.e., \$2.3/0.3). Here again, different percent values can be used for labor charges as a component of the total cost.

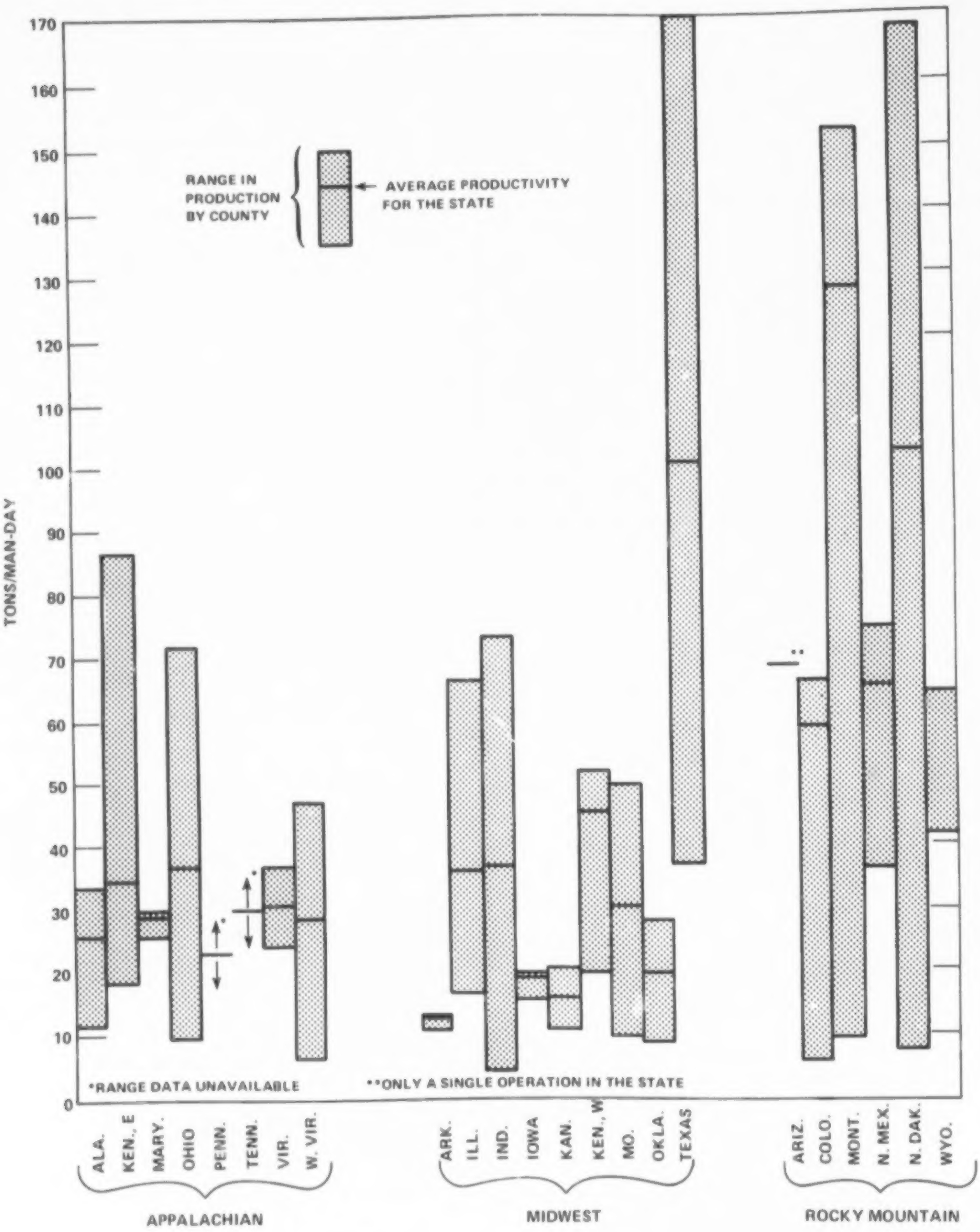

Figure 126. Productivities for surface coal mine operations in the United States, 1973.

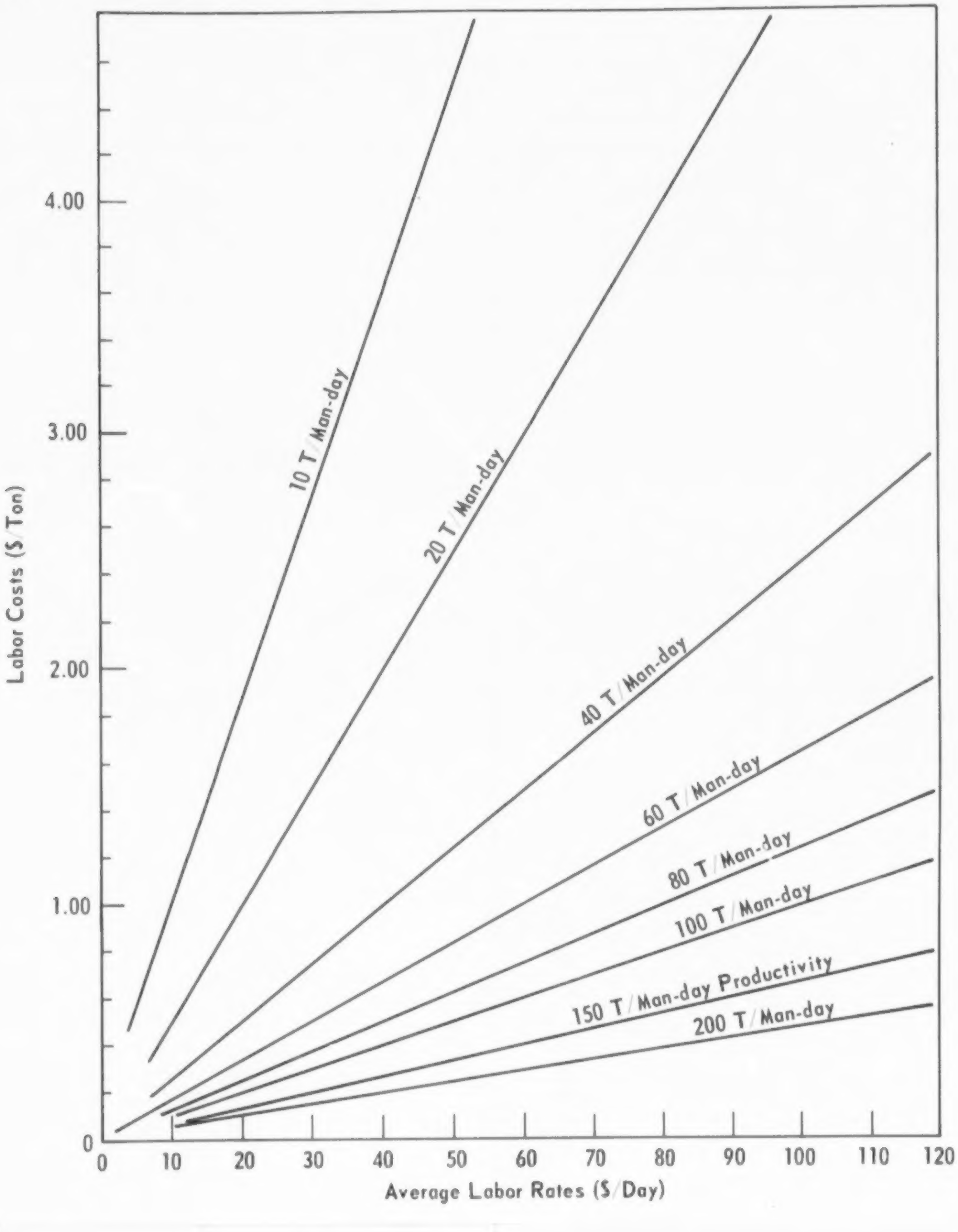

Figure 127. Graph for the calculation of labor costs per ton.

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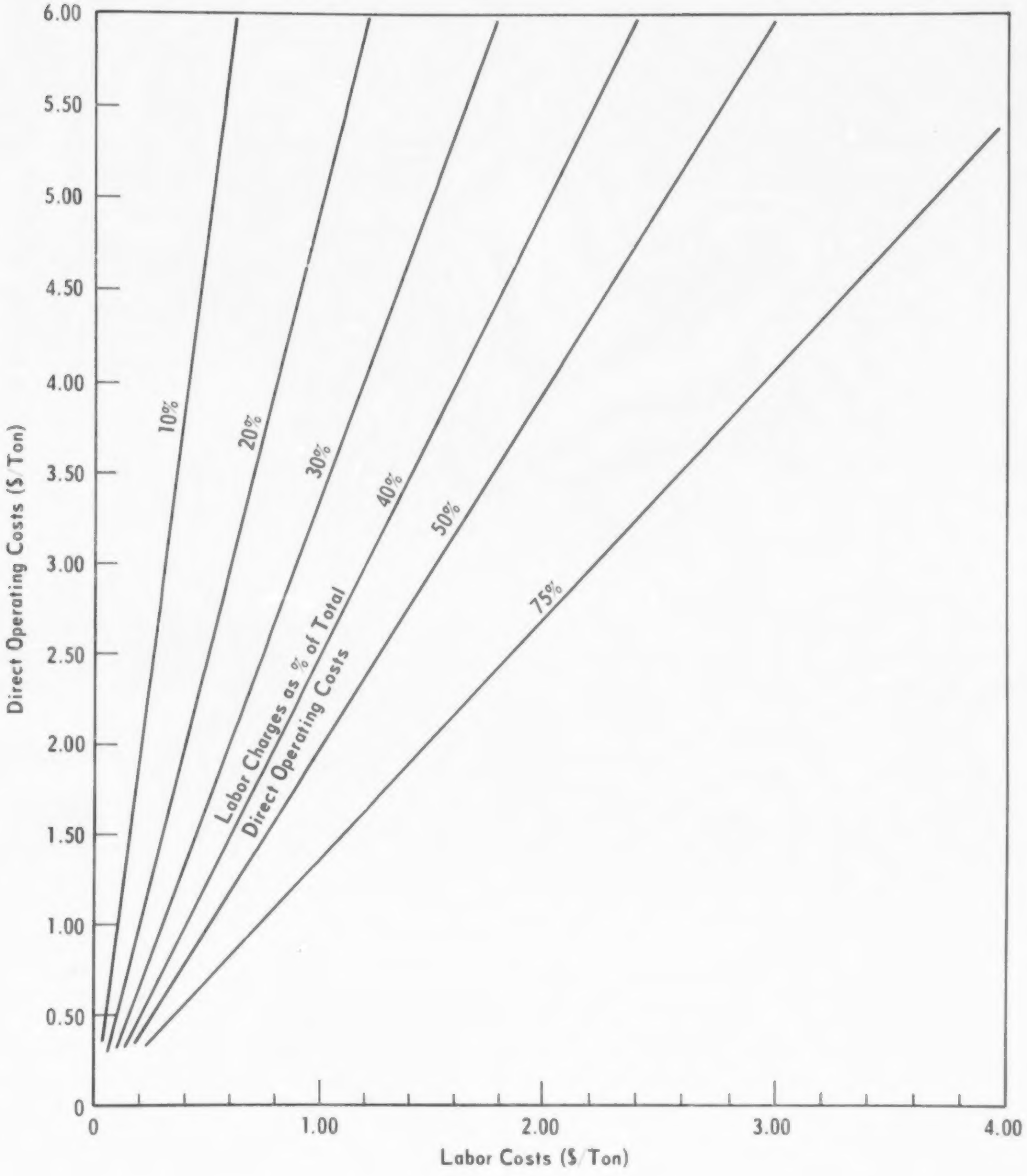

Figure 128. Relationship of labor costs and direct operating costs.

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The method of mine financing has an important bearing on total mining costs as does the schedule of depreciation. Figure 129 shows a range of typical debt-equity ratios commonly employed by the utility and mining industries. The lines below the 50/50 line are simply the inverse of those above and are added essentially to allow full graphical application in determining mining costs.

The start-up mine investment number derived in Figure 124 provides the basis for financing as it is assumed that equipment replacements and the like will be purchased from internally generated funds. This approach results in a steadily decreasing debt/equity ratio throughout the property life and would tend to increase ultimate mining costs as the risk-taking investor will require a somewhat greater return on his exposed funds. A mitigating circumstance, though, is the fact that the remaining book value of the mine plant will approach zero over the operating life and so the equity base, on which the return is calculated, will be shrinking. The assumption in this analysis, then, is that the per ton financial charges during the life of the property will approximate those established by the capital structure at the time of mine start up.

The example shows a 75/25 debt/equity ratio which may be occasioned through utility ownership and development of the reserves. The debt and equity components of the initial mine investment are $36,000,000 and $12,000,000, respectively.

From Figure 130, one can determine the financing charges per ton of coal depending on a choice of interest rates. Dividing the resulting numbers from Figure 130 by the annual productive capacity will allow direct determination of these charges. The calculations are illustrated here because the values for the example mine fall outside the limits of the graph.

Capital structure/annual ton is calculated as shown below.

Total capital investment = $73 million
Start up investment = $48 million
Debt component of start up investment = $36 million
Equity component of start up investment = $12 million

Capital structure (Debt component) per annual ton $= \frac{\$36 \text{ million}}{\$1.67 \text{ m tons}} = \$21.56$

Capital structure (Equity component) per annual ton $= \frac{\$12 \text{ million}}{\$1.67 \text{ m tons}} = \$\ 7.19$

Financing charges debt at 10% = $21.56 x 0.10= $ 2.16

Return on equity at 18% = $ 7.19 x 0.18= $ 1.29

Income taxes = 0.33 x $1.29 = $ 0.43

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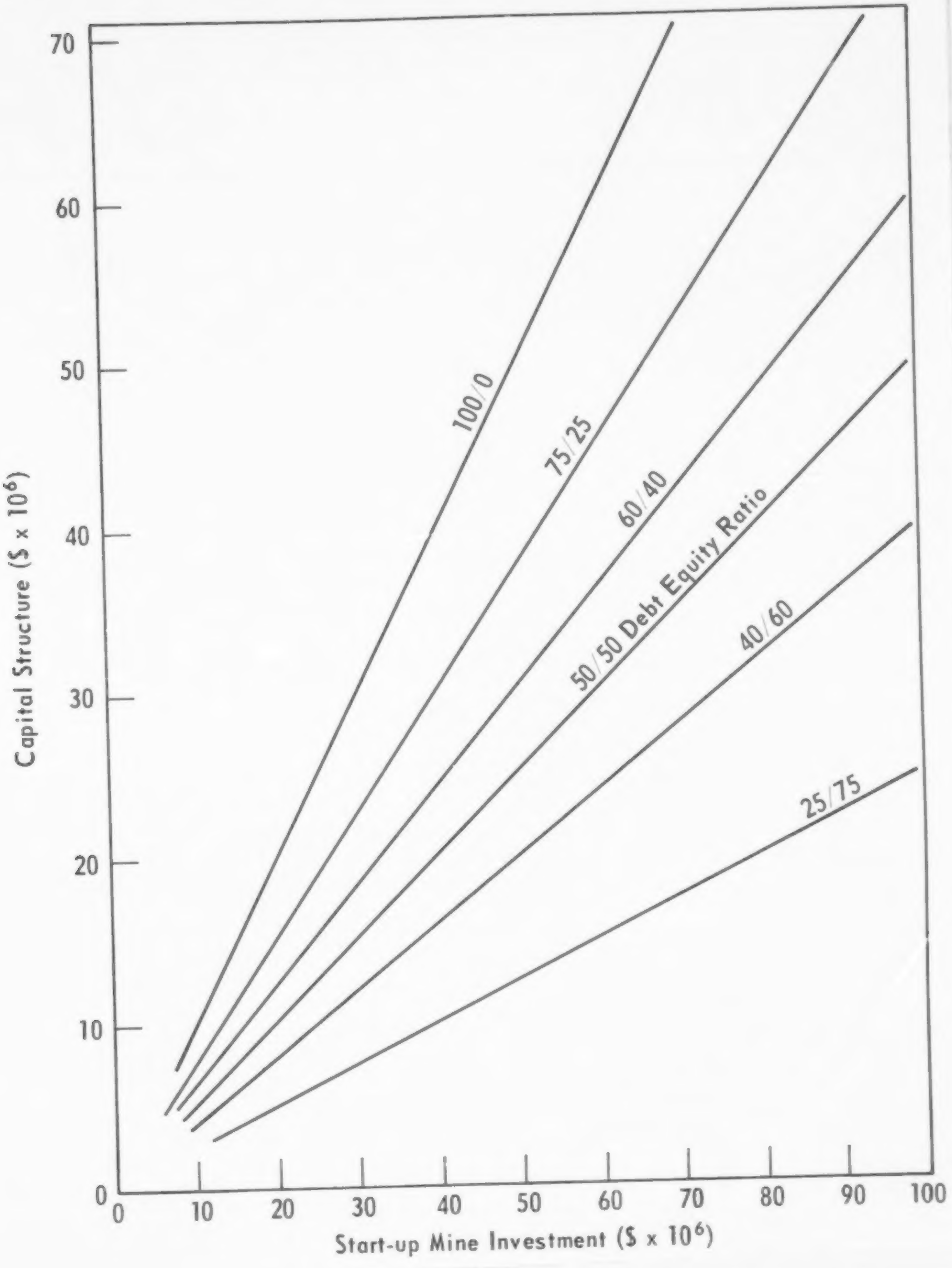

Figure 129. Capital structure for start-up mine investment.

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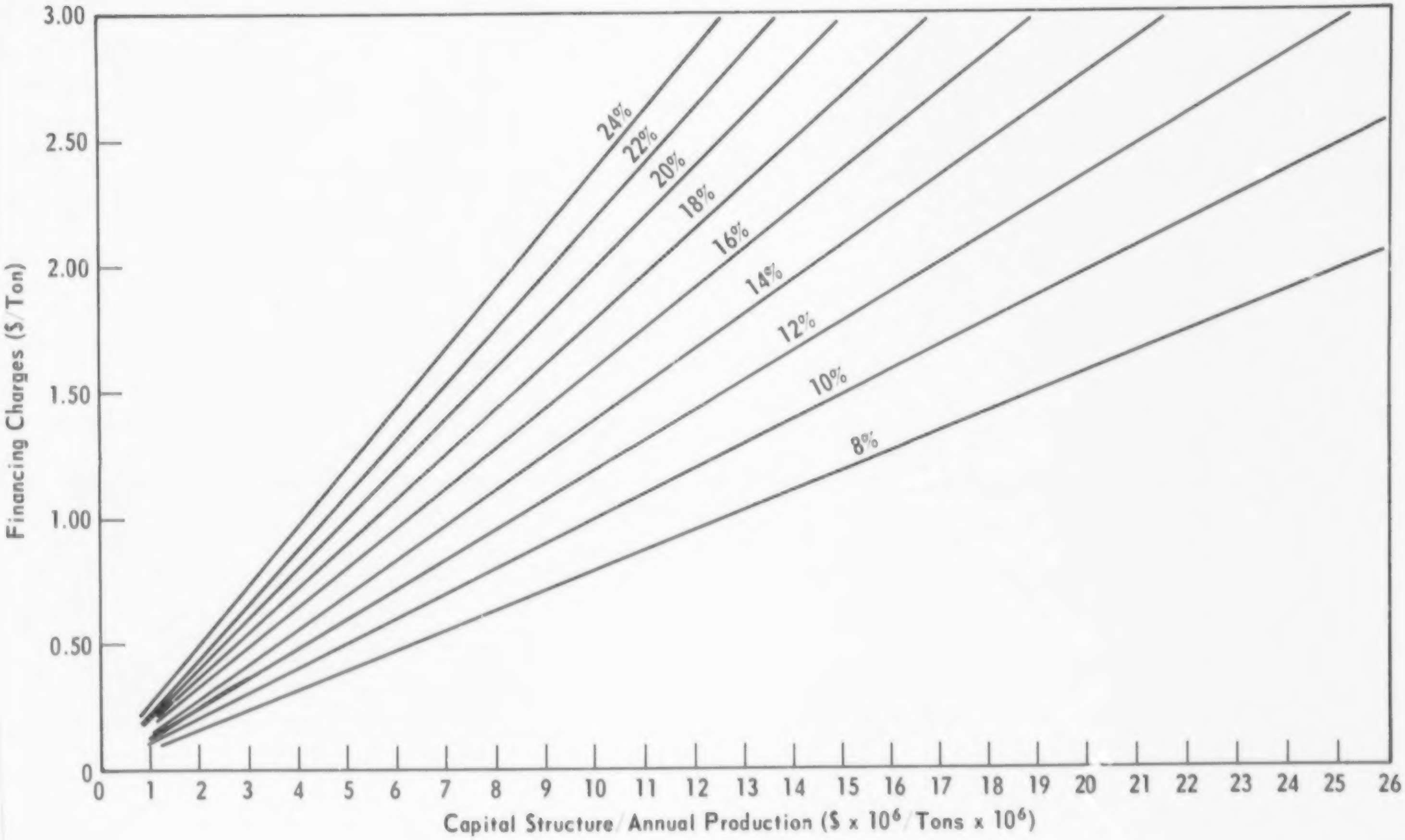

Figure 130. Graph for the calculation of financing charges and rate of return.

Thus assuming in the example a debt servicing rate of 10% and a return to equity of 18%, the burden will come to $2.12/tonne and $1.27/tonne ($2.16/ton $1.29/ton), respectively. One additional value can be calculated from Figure 130, the approximate federal and state income taxes. If the combined tax rate can be taken as 50%, and if one assumes a depletion allowance will be limited by the 50% of net income ruling, income taxes will total 33% of the return to equity figure. As illustrated in the example, these are $0.42/tonne ($0.43/ton).

Table 61 is the culmination of the graphical exercise wherein the previously determined charges from Figures 125, 128 and 130 are totalled to arrive at the minimum acceptable selling price per ton necessary to allow the mine operator an appropriate return on investment.

Referring once more to the example, the selling price for the particular operating and deposit characteristics should approximate $12.80/tonne ($13.01/ton).

SUMMARY

There are, of course, other approaches to costing an operation. For example, one major mining company has identified optimum mine sizes and mining methods on the basis of geographical and market conditions. It has economically optimized these selections through analysis of past practice. When the marketing department notifies engineering that it has received a contract, the engineering department selects an appropriate property which is held by the company in the geographical vicinity of the contract or at least as close to the contract delivery point as is possible to meet the type of product required. For the contract size specified, e.g., a five-million ton per year contract, the engineering department has, on file, the plans for mining development at the particular geographical location. Such an approach may be desired by large companies which can exercise economies of scale by utilizing the same, or closely related, approaches and equipment throughout their operations and are willing to sacrifice some potentially better approaches. For the average coal mining company, the sizes and types of deposits will vary over a large scale. Therefore, the design plans and economic analyses to determine the optimum, ultimate mine design will proceed in the more traditional fashion. As already indicated and referenced, computer-oriented cost analysis models are available for costing total surface mine operations or even individual unit operations.

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TABLE 61. CALCULATION OF SELLING PRICE

Depreciation cost/ton	(from Figure 125 or calculation)		$ 1.46
Direct operating cost/ton	(from Figure 128 or calculation)		$ 7.67
Debt financing cost/ton	(from Figure 130 or calculation)		$ 2.16
Return on investment	(from Figure 130 or calculation)		$ 1.29
Taxes @33% ROI	(from calculation)		$ 0.43
Minimum acceptable selling price			$13.01/ton
Start-up mine investment	(from Figure 124)	$48,000,000	
Total mine investment	(from Figure 124)	$73,000,000	
Mine capacity	(assumption)	1,670,000 tons	
Productivity	(assumption)	30 tons/man-day	
Personnel requirements: labor and supervisory (not including provisions for absenteeism, vacations, and turnover)	(calculation)	223 employees	

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EPA-287 (Cin.)
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SECTION 11

REFERENCES

Adams, L. M., Capp, J. P., Eisentrout, E., Reclamation of Acidic Coal Mine Spoil with Fly Ash, U.S. Department of the Interior, Bureau of Mines Report of Investigation Number 7504, April 1971.

Andrews, A. B., "Air Blast and Ground Vibrations in Open-Pit Mining," Mining Congress Journal, Vol. 61, No. 5, 1975.

Ash, R. L., "The Mechanics of Rock Breakage," Parts I through IV, Pit and Quarry, Vol. 56, Nos. 2, 3, 4, and 5, 1963.

Ash, R. L., "Cratering and Its Application in Blasting," Unpublished report, Department of Mining and Civil Engineering, University of Minnesota, 1970.

Atkinson, T., "Selection of Open-pit Excavating and Loading Equipment," Institution of Mining and Metallurgy, Vol. 80, Bulletin No. 776, Sec. A, 1971.

Bailey, P. A., "Exploration Methods and Equipment," Chapter 2.1, Surface Mining, Ed. E. P. Pfleider, SME/AIME, New York, 1968.

Bergmann, O. R., Wu, F. C., and Edl, J. W., "Model Rock Blasting Measures Effect of Delays and Hole Patterns on Rock Fragmentation," Engineering and Mining Journal, Vol. 175, No. 6, 1974.

Boddorff, D., "That's the Way Rock Breaks," The Explosives Engineer, Spring, 1974.

Boyce, T. A., "Economic System Analysis of Large Scale Surface Mining," Proceedings of the 3rd Symposium on Surface Mining and Reclamation, Vol. II, NCA/BCR, Washington, D.C., 1975.

Burton, A. K., "Off-Highway Trucks in the Mining Industry," Mining Engineering, Vol. 27, No. 8, 1975; Vol. 27, No. 10, 1975; Vol. 27, No. 11, 1975; Vol. 27, No. 12, 1975; Vol. 28, No. 1, 1976.

Caterpillar Tractor Co., Fundamentals of Earthmoving, Peoria, Illinois, July 1975.

Caterpillar Tractor, Co., Performance Handbook, 6th Edition, Peoria, Illinois, 1976.

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BOTTOM OF IMAGE AREA OUTSIDE DIMENSION FOR TABLES AND ILLUSTRATIONS
PAGE NUMBER
EPA-287 (Cin.) (4-78)

Cheek, E. E., "Application of a Standard Wheel Excavator in Coal," Mining Congress Journal, Vol. 52, No. 11, November 1966, pp. 33-35, 38-39.

Civil Engineering Handbook, 4th Edition, McGraw Hill Co., New York, 1962.

Condon, J. L., and Snodgrass, J. J., "Effects of Primer Type and Borehole Diameter on AN-FO Detonation Velocities," Mining Congress Journal, Vol. 60, No. 6, 1974.

Cummins, A. B., Chairman of the Editorial Board, SME Mining Engineering Handbook, AIME, New York, 1973.

Daley, D. D., "Emphasizing Environment - Natural Beauty Considered," Mining Engineering, Vol. 26, No. 2, 1974.

Dick, R. A., "Explosives and Borehole Loading," SME Mining Engineers Handbook, SME/AIME, New York, 1973.

Dombrowsky, N. G., "The New Bucket Wheel Excavator and Its Construction Problems," Bergbautechnik, Vol. 14, 1964, p. 277.

Doyle, F. J., "Variables Affecting Grading Costs for Strip Mine and Refuse Bank Reclamation," Proceedings of the First Symposium on Mine and Preparation Plant Refuse Dispoal, National Coal Association, 1974, pp. 212-227.

Drevdahl, E. R., Profitable Use of Excavation Equipment, Technical Publications, Desert Laboratories, Inc., Tuscon, Arizona, 1961.

DuPont Co., Blaster's Handbook, Wilmington, Deleware, 1967.

Ellison, R. D., and Thurman, A. G., "Geotechnology: An Integral Part of Mine Planning," A Chapter in Coal Exploration, Ed. W. G. Muir, Miller Freeman Publications, Inc., San Francisco, 1976.

EPRI, Coal Mining Cost Models - Surface Mines, Electric Power Research Institute, EPRI EA-437, Research Project 435-1, 1977.

Felix, G. L., "Development of the 350 Ton Haulage Truck," Mining Congress Journal, Vol. 62, No. 3, 1976.

Gartner, K., "Bucket Wheel Excavator on Building Sites," Forden und Heben, Vol. 3, No. 6, November 1965.

Gordon, R. L., Historical Trends in Coal Utilization and Supply, U.S. Bureau of Mines Report, Grant No. G0155116, Washington, D.C., 1976.

Grim, E. C., and Hill, R. D., Environmental Protection in Surface Mining of Coal, Report No. EPA-670/2-74-093, U.S. Environmental Protection Agency, Cincinnati, Ohio, 1974.

Haley, W. A., "Changing Methods and Equipment Use in Appalachian Coal Mining," Second Research and Applied Technology Symposium on Mined Land Reclamation, NCA/BCR, 1974.

Highway Engineering Handbook, 1st Edition, McGraw Hill Co., New York, 1960.

Hill, R. D., and Grim, E. C., "Environmental Factors in Surface Mine Recovery," Symposium on Restoration and Recovery of Damaged Ecosystems, VPI & SU, Blacksburg, Virginia, March 1975.

Himmel, W., "The Specific Digging Resistance in Dependency on the Shaving Area and Shaving Form in Different Soils," *Freiberger Forschungshefte*, A 265, 1961, pp. 1-40.

International Harvester, "An Analysis of Equipment Selection for Heavy Construction," Payline Division, 1975.

Katell, S., and Hemingway, E. L., "Cost Analysis of Model Mines for Strip Mining of Coal in the United States," *U.S. Bureau of Mines, Information Circular* 8535, 1972.

Katell, S., Hemingway, E. L., and Berkshire, L. H., "Basic Estimated Capital and Operating Costs for Coal Strip Mines," Revision of IC 8661, *U.S. Bureau of Mines, Information Circular* 8703, 1976.

Kimball, L. Robert (Consulting Engineers), *Slope Stability, Volume 1 - Report and Field Book*, Report ARC-71-66-T3 prepared for the Appalachian Regional Commission, p. 81, undated.

Krause, R. R., "Predicting Mined Land Soil," Chapter in *Ecology and Reclamation of Devastated Land*, Vol. 1, Ed., R. J. Hutnik and Grant Davis, Gordon Breach, New York, 1973.

Learmont, T., "Productivity in Large Stripping Machines," *SME/AIME Transactions*, Vol. 258, No. 3, 1975.

Maneval, D. R., "Coal Mining vs. Environment: A Reconciliation in Pennsylvania," *Appalachia, A Journal of the Appalachian Regional Commission*, Vol. 5, No. 2, February-March 1972.

Mani, M. S., "Design Features of Bucket Wheel Excavators," *Journal of Mines, Metals and Fuels*, Vol. 14, No. 3, March 1966, pp. 71-79.

Manula, C. B., Albert, E. K., and Burgos, V., *A Report on Anthracite Open-pit Mining, Part V. Engineering and Mine Cost Analysis*, Special Research Report SR-104, Coal Research Section, The Pennsylvania State University, 1976.

Mathematica, Inc., *The Design of Surface Mining Systems in Eastern Kentucky*, Department of Natural Resources and Environmental Protection, Division of Reclamation, April 30, 1973.

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EPA-287 (Cin.)
(4-76)

McGraw Hill Inc., Coal Age, Vol. 75, No. 7, 1970, pp. 173-179.

MESA, Engineering and Design Manual: Coal Refuse Disposal Facilities, Prepared by E. D'appolonia Consulting Engineer for U.S. Department of the Interior, Mining Enforcement and Safety Administration.

Meyer, C. F., Route Surveying, International Textbook Co., Scranton, PA, 1969.

Morgenstern, N. R., and Price, V. E., "The Analysis of the Stability of General Slip Surfaces, " Geotechnique, Vol. 15, No. 1, pp. 79-93, 1965.

Morrell, R. J., and Unger, H. F., "Drilling Machines, Surface," SME Mining Engineering Handbook, SME/AIME, New York, 1973.

Murray, C. W., "A Computer Model for Optimizing Surface Mine Haul Road Profile," M. Eng. Report, Mineral Engineering Management, The Pennsylvania State University, 1978.

Nicholls, H. R., Johnson, C. J., and Duvall, W. I., Blasting Vibrations and Their Effects on Structures, Bulletin 856, U.S. Department of the Interior, Bureau of Mines, 1971.

Nordness, H. J., "Horizontal Drills for Surface Mines," Proceedings 3rd Conference on Mine Productivity, The Pennsylvania State University, 1976.

Obert, L. A., and Duvall, W. I., Rock Mechanics and the Design of Structures in Rock, John Wiley, New York, 1967.

Otte, J. A., and Boehlje, M., "A Model to Analyze the Cost of Strip Mining and Reclamation," Proceedings of the 3rd Symposium on Surface Mining and Reclamation, Vol. II, NCA/BCR, Washington, D.C., 1975.

Peurifoy, R. L., Construction Planning, Equipment and Methods, McGraw Hill, New York, 1970.

Pfleider, E. P., Ed., Surface Mining, American Institute of Mining, Metallurgical and Petroleum Engineers, Inc., New York, 1968.

Porter, W. E., "Multiple Seam Strip Mining: A Survey of Economic Feasibility Model," Unpublished M. Eng. Report, The Pennsylvania State University, 1971.

Price, G. C., Manula, C. B., and Ramani, R. V., Material Handling Research, The Bucket Wheel Excavator, U.S. Bureau of Mines, Information Circular 8580, 1973.

Pugliese, J. M., Designing Blast Patterns Using Empirical Formulas, U.S. Bureau of Mines, Information Circular 8550, Washington, D.C., 1972.

Ramani, R. V., "Computer Simulation of a Bucket Wheel Excavator," Unpublished M.S. Thesis, The Pennsylvania State University, 1968.

BOTTOM OF IMAGE AREA OUTSIDE DIMENSION FOR TABLES AND ILLUSTRATIONS
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EPA-287 (Cin.)
(4-76)

Ramani, R. V., and Grim, E. C., "Surface Mining-A Review of Practices and Progress in Land Disturbance Control," Chapter 14 in the book *Reclamation of Drastically Disturbed Lands*, Ed., F. W. Schaller and P. Sutton, American Society of Agronomy, Madison, Wisconsin, 1978.

Ramani, R. V., Murray, C. W., and Manula, C. B., *Application of a Total System Surface Mine Simulator to Coal Stripping, Volume IV, Surface Mine Cost Model*, National Technical Information Service (NTIS), Publication No. 294634/AS, 1979.

Rasper, L., and Ritter, H., "Opening Up of the Lignite Mine of the Neyveli Lignite Corporation and Experience with Bucket Wheel Excavators in Hard Overburden," *Braunkohle, Warme und Energie*, Vol. 13, No. 10, October 1961, pp. 390-400.

Rasper, L., "Present Stage of the Development of Bucket Wheel Excavators in the German Federal Republic," *Braunkohle, Warme und Energie*, Vol. 16, No. 11, November 1964, pp. 457-471.

Rumfelt, H., "Computer Methods for Estimating Proper Machine Mass for Stripping Overburden," *Mining Engineering*, Vol. 13, No. 5, 1961.

Rumfelt, H., "Application and Performance of Wheel Excavators," *Mining Congress Journal*, Vol. 47, No. 6, June 1961, pp. 46-49.

Saperstein, L. W., and Secor, E. S., "Improved Reclamation Potential with the Block Method of Contour Stripping," Proceedings of the Research and Applied Technology Symposium on Mined-Land Reclamation, Pittsburgh, Pennsylvania, Bituminous Coal Research, Inc., NCA/BCR, March 1973.

Scott, R. F., *Principles of Soil Mechanics*, Addison-Wesley, Reading, Massachusetts, 1963.

Secor, E. S., "An Economic Comparison Between the Block and Conventional Methods of Contour Strip Mining," Unpublished M. Eng. report in Mineral Engineering Management, The Pennsylvania State University, 85 pp., 1977.

Skelly and Loy Engineering Consultants, *Economic Engineering Analysis of U.S. Surface Coal Mines and Effective Land Reclamation*, U.S. Bureau of Mines Report, Contract No. S0241049, Washington, D.C., 1975.

Starr, C., "Energy and Power," *Scientific American*, Vol. 225, No. 3, September 1971, pp. 37-49.

Stefanko, R., Ramani, R. V., and Ferko, M. R., *An Analysis of Strip Mining Methods and Equipment Selection*, Research and Development Report No. 61, Interim Report No. 7, Prepared by Coal Research Section, The Pennsylvania State University, for Office of Coal Research, U.S. Department of the Interior, Washington, D.C., May 1973.

Terzaghi, K., and R. B. Peck, Soil Mechanics in Engineering Practice, 2nd Edition, John Wiley, New York, 1967.

Thomas, L. J., Introduction to Mining, Hicks and Sons, Sydney, Australia, 1973.

Toth, G. W., and Stinnett, L. A., "Techniques of Rapid Preliminary Estimates of Coal Mining Costs," paper presented at the 1976 Mining Convention, American Mining Congress, Denver, Colorado, September 26-29, 1976.

U.S. Department of the Interior, Surface Mining and Our Environment, A Special Report to the Nation, 1967.

Wang, F. D., et al., Computer Program for Pit Slope Stability Analysis by the Finite Element Stress Analysis and Limiting Equilibrium Methods, U.S. Bureau of Mines, Report of Investigations 7685, p. 53, 1972.

Wendertz, R. L., "Coal Loading Shovels," (a review of factors to consider in loading equipment selection), Mining Congress Journal, Vol. 56, No. 4, 1970.

SECTION 12

BIBLIOGRAPHY

Adams, L. M., Capp, J. P., Gillmore, D. W., "Coal Mine Spoil and Refuse Bank Reclamation with Powerplant Fly Ash," Proceedings of the Third Mineral Waste Utilization Symposium, Chicago, Illinois, March 1972.

Alder, L., and Nauman, H. E., *Analyzing Excavation and Material Handling Equipment*, VPI, Blacksburg, Virginia, February 1970.

Aiken, G. E., "Bucket Wheel Excavators: How to Choose the Right One for the Job," *Mining Engineering*, Vol. 18, No. 1, January 1966, pp. 71-79.

AIME, "Drilling Selection Requires Value Judgement," *Mining Engineering*, Vol. 19, No. 10, 1967.

AIME, "Excavation and Loading - A Job for the Giants," *Mining Engineering*, Vol. 19, No. 10, 1967.

Allen, N., "Experimental Multiple Seam Mining and Reclamation on Steep Mountain Slopes," Proceedings of the Research and Applied Technology Symposium on Mined-Land Reclamation, Pittsburgh, Pennsylvania, NCA/BCR, March 1973.

American Iron and Steel Institute, *Handbook of Steel Drainage and Highway Construction Products*, 2nd Edition, New York, 1971.

Archibald, J. C., "Application of Horizontal Drilling as a Replacement of Vertical Drilling Techniques," *Joy Manufacturing Company*, Pittsburgh, Pennsylvania, 1976.

Ash, R. L., "The Design of Blasting Rounds," *Surface Mining*, Ed. E. P. Pfleider, SME/AIME, New York, 1968.

Atkinson, T., and Pryor, R., "Selection of Open-Pit Excavating and Loading Equipment discussion on," *Institution of Mining and Metallurgy*, Vol. 80, Bulletin No. 777, Sec. A, 1971.

Battelle Columbus Laboratories, *Energy from Coal: Guidelines for the Preparation of Environmental Impact Statement*, NTIS Publication PB-242-960, April 1975.

Bellum, D. P., "Open-Pit Mining," *Mining Engineering*, Vol. 28, No. 2, 1976.

Bengston, G. W., Mays, D. A., and Allen, J. C., "Revegetation of Coal Spoil in Northeastern Alabama: Effects of Timing of Seeding and Fertilization on Establishment of Pine-Grass Mixtures," Proceedings of the Research and Applied Technology Symposium on Mined-Land Reclamation, Pittsburgh, Pennsylvania, NCA/BCR, March 1973.

Burgos, V. T., "A Study on the Feasibility of Open-Pit Anthracite Mining," Unpublished M. Eng. Report, The Pennsylvania State University, 1975.

Caruccio, F. T., "An Evaluation of Factors Affecting Acid Mine Drainage Production and the Ground Water Interactions in Selected Areas of Western Pennsylvania," Second Symposium on Coal Mine Drainage Research, Pittsburgh, Pennsylvania, NCA/BCR, May 1968.

Day, D. A., Construction Equipment Guide, A. Wiley & Sons, New York, 1974.

Davis, G., Strip-Mine Reclamation in Appalachia (Review Draft), U.S. Department of Agriculture, Forest Service, Northeastern Forest Experiment Station, July 1971.

Dawson, V. E., "Observations Concerning On Site Brake Testing of Large Mining Trucks in British Columbia," SAE Paper 75060, Society of Automotive Engineers, Warrendale, Pennsylvania, 1975.

Dotson, G. K., Constraints to Spreading Sewage Sludge on Croplands, U.S. Environmental Protection Agency, Cincinnati, Ohio, 1973.

Duggan, C., Scanlon, D. H., and Bean, S. D., "Evaluation of Municipal Compost for Strip Mine Reclamation," Compost Science; Journal of Waste Recycling, Vol. 14, No. 3, May-June 1973.

Environmental Protection Agency, Guidelines for Erosion and Sediment Control Planning and Implementation, Environmental Protection Technology Series, EPA-R2-72-015 August 1972, [Project 15030 FMZ. Prepared by Department of Water Resources, State of Maryland, and Hittman Associates, Inc., Columbia, Maryland].

Enyedy, G., Jr., "AACE's Cost of Major Equipment (COME) Project in the Mineral Industry," AIME Council of Economic Proceedings, 1971.

Fites, D. V., "Tractor Scrapers Break New Ground," Mining Engineering, Vol. 21, No. 5, May 1969, pp. 69-70.

Ford, W. W., "Slope Reduction Methods," Unpublished Report, Division of Reclamation, Department of Natural Resources, Frankfort, Kentucky, 1968.

Ford, W. W., "Slope Reduction Method (Parallel Fill)," Unpublished Report, Division of Reclamation, Department of Natural Resources, Frankfort, Kentucky, 1970.

Gartner, E., "Trends of Development in Mining and Haulage Equipment of the Rhenish Brown Coal Open Cut Mines," *Braunkohle Warme und Energie*, Vol. 7, Nos. 11 and 12, June 1955, pp. 226-241.

Grant, A. F., and Lang, A. L., *Reclaiming Illinois Strip Coal Land with Legumes and Grasses*, Bulletin 628, University of Illinois, Agriculture Experiment Station, Urbana, Illinois, 1958.

Griffith, F. E., Magnuson, M. S., and Kimball, R. F., *Demonstration and Evaluation of Five Methods of Secondary Backfilling of Strip-Mine Areas*, U.S. Department of the Interior, Bureau of Mines, Report of Investigations No. 6772, 1966.

Grube, W. E., Jr., Smith, R. M., Singh, R. and Sobek, A. A., "Characterization of Coal Overburden Materials and Proceedings," Research and Applied Technology Symposium on Mined-Land Reclamation, Pittsburgh, Pennsylvania, NCA/BCR, March 1973.

Gwinn, T. A., "An Improved Environment Through Intelligent Mined-Land Reclamation," Paper presented at the fall meeting of the Institute of Mining, Metallurgical, and Petroleum Engineers, Inc., Salt Lake City, Utah, September 17 to 19, 1969, pp. 28-30.

Hartman, H. L., Principles of Drilling, Chapter 6.1, *Surface Mining*, Ed. E. P. Pfleider, SME/AIME, New York, 1968.

Heine, W. N., Gukert, W. E., "A New Method of Surface Coal Mining in Steep Terrain," Proceedings of the Research and Applied Technology Symposium on Mined-Land Reclamation, Pittsburgh, Pennsylvania, NCA/BCR, March 1973.

Hewes, L. I., and Oblesby, C. H., *Highway Engineering*, John Wiley and Sons, New York, 1960.

Hinesly, T. D., Braids, O. C., and Molina, J. E., *Agricultural Benefits and Environmental Changes Resulting from the Use of Digested Sewage Sludge on Field Crops*, U.S. Environmental Protection Agency (SW-30d), Cincinnati, Ohio, 1971.

Hinesly, T. D., Jones, R. L., and Sovewitz, B., "Use of Waste Treatment Plant Solids for Mined Land Reclamation," *Mining Congress Journal*, September 1972.

Hollingsworth, J. A., "History of Development of Strip Mining Machines," *Bucyrus-Erie Company Publication*, pp. 1-16.

Holzman, R. S., *Executive Tax Management*, Thomas V. Cromwell Co., New York, 1974.

Howarth, G. R., "How Will Major New Mines be Financed in the Future," SME Preprint No. 77-K-125, Paper presented at the AIME-Annual Meeting, Atlanta, 1977.

Hubbert, M. K., "The Energy Resources of the Earth," Scientific American, Vol. 225, No. 3, September 1971, pp. 61-70.

Jarpa, S. G., "Capital Investment and Operating Cost Estimation in Open-Pit Mining," Application of Computer Methods in the Mineral Industry, Ed. R. V. Ramani, Society of Mining Engineers of AIME, Salt Lake City, 1977.

Jelen, F. C., Editor, Cost and Optimization Engineering, McGraw Hill Inc., New York, 1970.

Kentucky Department for Natural Resources and Environmental Protection, A Manual of Kentucky Reclamation, Frankfort, Kentucky, May 1973.

Killebrew, C. E., "Tractor Shovels, Tractor Dozers, Tractor Scrapers," Surface Mining, Pfleider, Eugene P. (Ed.), The American Institute of Mining, Metallurgical, and Petroleum Engineers, Inc., New York, 1968, pp. 463-477.

Krumrey, A., "Krupp Bucket Wheel Excavators with Stacker Boom," Krupp Technical Review, Vol. 23, No. 1, 1965.

Levene, H. D.,"An Unusual Coal Mine/Power Plant Complex," Coal Mining and Processing, October, 1971.

Limstrom, G. A. and Deitschman, G. H., "Reclaiming Illinois Strip Coal Lands by Forest Planting," University of Illinois Bulletin, 547, 1951, p. 50.

Linden, G., "The Bucket Wheel Excavator and the Versatility in Practice," Krupp Technical Publication. Also published in Deutsche Hebeword Fordertechnik, Nos. 3 and 4, July and October 1957.

Loftis, J. B., "The Evaluation of Horizontal Drilling In Open-Pit Mining," SME/AIME Preprint No. 75-AO-319, 1975.

Loper, F. G., "Simulation Techniques Pay Their Way as Management Tools," Mining Engineering, Vol. 19, No. 1, 1967.

Lyon, T. L., Buckman, H. O., and Brady, N. C., The Nature and Properties of Soil, 5th Ed., MacMillian Company, New York, 1952.

Mackellar, A. J., "Cost Engineering - Who Needs It?" SME Preprint 75-B-23, Paper presented at the AIME Annual Meeting, New York, 1975.

Manula, C. B., Albert, E. K., and Burgos, V., A Report on Anthracite Open-Pit Mining Part II. Engineering and Mine Cost Analysis, Special Research Report SR-104, Coal Research Section, The Pennsylvania State University, 1976.

Manula, C. B., Albert, E. K., and Ramani, R. V., "Application of a Total Systems Simulator to Coal Stripping, Volume II: OPMHS User's Manual," U.S. Bureau of Mines, 1977.

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Manula, C. B., "OPMHS - Open-Pit Materials Handling Simulator," XIth APCOM Symposium, University of Arizona, 1973.

Maraboli, L. O., "Computer Simulation and Cost Analysis of Materials Handling in Chuquicamata, Chile," Unpublished M. Eng. Report, The Pennsylvania State University, 1973.

Marcus, J. J., "The Buckingham Method - An Aid in Equipment Selection," Mining Engineering, Vol. 17, No. 9, September 1965, pp. 92-94.

McClay, J., "A Dynamic Costing Model for Mining Systems," Unpublished M.S. Thesis, The Pennsylvania State University, 1974.

McFadden, D. H., "Pit Planning and Design-Coal Mines," Surface Mining, Pfleider, Eugene P. (Ed.), The Institute of Mining, Metallurgical, and Petroleum Engineers, Inc., New York, 1968, pp. 211-216.

McGraw Hill Inc., Coal Age, Vol. 75, No. 7, 1970, pp. 173-179.

Morgan, W., Fleet Production and Cost Program, Caterpillar Tractor Co., Peoria, Illinois, 1975.

Nalle, P. B., and Week, L. W., "The Digital Computer - Application in Mining and Process Control," Mining Engineering, Vol. 28, No. 9, 1976.

National Coal Association, Mined-Land Conservation Conference, Vol. 7, No. 4, Washington, D.C., November-December, 1971.

Nutter, F. B., "Mining with Front-End Loaders," Mining Congress Journal, Vol. 56, No. 6, June 1970, pp. 60-64.

Olivares, L. V., "Theory, Research and Practical Results of Inclined Drilling," An Unpublished Report in Mineral Engineering Management, The Pennsylvania State University, December 1967.

O'Neil, T. J., "Computer Simulation of Materials Handling in Open-Pit Mines," Unpublished M.S. Thesis, The Pennsylvania State University, 1966.

O'Neil, T. J., "The Minerals Depletion Allowance: Its Importance in Nonferous Metal Mining," Mining Engineering, Vol. 26, No. 10, 1974.

Peabody Coal Company, "A Model Mining Company," Coal Age, Vol. 76, No. 10, 1971.

Perez, R., "Application of a Time-Oriented Total Systems Simulator for Open-Pit Mining," Unpublished M. Eng. Report, The Pennsylvania State University, 1973.

Peterson, J. R., and Gschwind, J., "Amelioration of Coal Mine Spoils with Digested Sewage Sludge," Proceedings of the Research and Applied Technology Symposium on Mined-Land Reclamation, Pittsburgh, Pennsylvania, NCA/BCR, March 1973.

Plass, W. T., and Capp, J. P., "Physical and Chemical Characteristics of Sulfur Mine Spoil Treated with Fly Ash," Journal of Soil and Water Conservation, Vol. 29, No. 3, 1974.

Ramani, R. V., "Environmental Concerns and Decisions for the Mining Industry," Journal of the Institution of Engineers (India), MM Vol. 54, No. 2, March 1974.

Ramani, R. V., "Surface Mining," Mining Engineering, Vol. 26, No. 2, 1974.

Ramani, R. V., Iroegbu, M., and Manula, C. B., "Application of a Total Systems Simulator to Coal Stripping, Volume III: Equipment Selection Models," U.S. Bureau of Mines, 1977.

Riley, C. V., "Design Criteria of Mined Land Reclamation," Paper presented at the SME Fall Meeting and Exhibit, Birmingham, Alabama, October 1972.

Riley, C. V., "Furrow Grading--Key to Successful Reclamation," Proceedings of the Research and Applied Technology Symposium on Mined-Land Reclamation, Pittsburgh, Pennsylvania, NCA/BCR, March 1973.

Rodgers, H. C. C., "Developments in Equipment Used for Overburden Removal and Coal Winning in the Brown Coal Industry. Excavating Equipment for Brown Coal, Part I," Proceedings of the Australian Institute of Mining and Metallurgy, No. 194, June 1960, pp. 127-249.

Rose, B. A., "Sanitary District Puts Sludge to Work in Land Reclamation," Water and Sewage Works, A Scranton Gillette Publication, September, 1968.

Sathit, T., "Principles of Drilling," Mining Engineering Handbook, Cumins and Given, SME, Mudd Series, New York, 1973.

Sanford, R. L., "Stochastic Simulation of a Belt Conveyor System," Unpublished M.S. Thesis, The Pennsylvania State University, 1965.

Schellhorn, H. W., "Some Aspects of High Capacity Production with Bucket Wheel Excavators," Krupp Technical Publication. Also published in The K. A. Clark Volume, Research Council of Alberta, Edmonton, October 1963.

Schmidt, R. A., and Stoneman, W. C., A Study of Surface Coal Mining in West Virginia, Stanford Research Institute, Menlo Park, California, 1972.

Scott, R. B., Hill, R. D., and Wilmoth, R. C., Cost of Reclamation and Mine Drainage Abatement--Elkins Demonstration Project, U.S. Environmental Protection Agency, Cincinnati, Ohio, 1970.

Skelly and Loy Engineering Consultants, Surface Mine Haul Road Design Study, U.S. Bureau of Mines Report, Contract No. J0255037, Washington, D.C. 1976.

Society of Mining Engineers of AIME, Mining Engineering Handbook, New York, 1973.

Staff, Surface Mining of Coal, Coal Age Guidebook, July 1970.

Stermole, F. J., Economic Evaluation and Investment Decision Methods, Investment Eval. Corp., Golden, Colorado, 1974.

Taylor, D. W., Soil Mechanics, John Wiley and Sons, New York, 1960.

Taylor, G. A., Managerial and Engineering Economy, V. Nostrand Reinhold Co., New York, 1964.

U.S. Department of Agriculture, Kentucky Guide for Classification, Use, and Vegetative Treatment of Surface Mine Spoil, Soil Conservation Service, Lexington, Kentucky, 1971.

Weimer, W. H., "Production and Engineering in Surface Coal Mines," Surface Mining, Pfleider, E. P. (Ed.), The Institute of Mining, Metallurgical, and Petroleum Engineers, Inc., New York, 1968, pp. 229-238.

Wimpfen, F. P., "Mine Costs and Control," Section 31, SME Mining Engineer's Handbook, AIME, New York, 1973.

Woodruff, S. D., Methods of Working Coal and Metal Mines, Planning and Operations, Vol. 3, Pergamon Press, Inc., New York, 1966.

Wurtz, C. B., "Mining and Conservation," Mining Congress Journal, Vol. 56, No. 5, May 1970, pp. 40.

Zemp, J. C., "Improved Blasting Techniques Give Wider Applications for Wheel Loaders," Mining Magazine, Vol. 121, No. 3, September 1969, p. 207.